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President, 1918-1919.

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Vol. XXIII.

JULY, 1922.

No. 1.

THE APPLICATION OF FLOTATION TO THE ANTIMONIAL GOLD ORES
OF THE MURCHISON RANGE.

By H. R. ADAM.

Some years ago the writer reported¹ experiments showing that in the flotation of the stibnite bearing ores of the Murchison Range, using hot acid solutions, the bulk of the gold was left with the tailing, *e.g.*, in samples of antimony content between three and twenty-eight per cent. Sb., the proportion of gold left with the tailing varied from sixty to eighty per cent. of the total. It was thought that these results might be the basis of a suitable treatment, and a few amalgamation tests on small quantities of ore, from which the stibnite had been removed by flotation, gave reasonably encouraging results. The method of flotation used in the initial experiments was the rather out of date one, in which a 1% or 2% H_2SO_4 solution at about 70°C. is used. This gives an excellent separation of the stibnite, although the addition of oil is required when the proportion of sulphide mineral is at all high; although probably not practically feasible, it is a useful method for investigation purposes, since much coarser material can be dealt with than is the case with the standard mechanical agitation method.

The details of one of the best of these preliminary trials were as follows:—

- (1) 200 grams unweathered ore crushed to pass a 40 mesh sieve:

Au content, 22 dwt. per ton.

Stibnite, 28%.²

Yielded 46 grams float @ 24.8 dwt. per ton gold and 148 grams tailing, which after grinding with mercury

in a mortar, gave a residue of 1.7 dwt. per ton.

In round figures, the proportion of the total gold in the tailing was 75%, so that the recovery by amalgamation was 69%.

A similar trial on partly oxidised ore, which occurs to a considerable extent near the surface, and is frequently very rich, gave the following figures:—

- (2) The sample, ground to pass a 30 mesh sieve, contained antimony as oxide and sulphide, also a little copper as sulphide and carbonate. The total sulphide may be taken as 12%, which was the percentage weight of the float.

Au in original sample, 129 dwt. per ton.

Per cent. weight of froth, 12; Au 98 dwt. per ton.

Per cent. weight of tailing, 80; Au 142 dwt. per ton.

The 8% unaccounted for comprised mechanical losses, and the Ca, Mg and Cu which went into solution in the weak acid.

The flotation tailing on amalgamation gave a residue of 42 dwt. per ton, so that 60% of the total gold was obtained.

The next material experimented with consisted of samples which were really concentrates obtained by passing rather coarsely crushed ore once over a small Wilfley table. In explanation of the rather curious procedure of trying to treat a "gravity" concentrate by flotation it may be said that some information regarding flotation was being sought, and also that such a treatment corresponds to the "cleaning" of a flotation

¹ *Jour. Chem. Met. and Min. Soc.*, June, 1915.

² Calculated from the antimony content.

concentrate, which is now common practice. As some useful deductions were made from these tests, the results may be summarised here. The two samples treated were of the same nature as before, *viz.*, one in which the antimony was present as fresh sulphide and one of partly weathered ore, containing both sulphide and oxide of antimony.

(3) Clean sulphide ore dry crushed according to the following grading:—

−20, 10%; +30, 19%; +50, 31%;
+150, 21%; −150, 19%.

Au 14.8 dwt. per ton.

Stibnite, 17%.

2,500 grams passed once over Wilfley table, yielding—

820 grams concentrate (33%). Au 36 dwt. per ton (79% of total Au); Stibnite, 40%.

1,500 grams sand tailing; Au 4.6 dwt. per ton (18% of total Au). Stibnite, 40%.

180 grams slime tailing. Au 6.0 dwt. per ton (3% of total Au); Stibnite, 40%.

The concentrate here really represents concentrate and "middling," as it was necessary to collect sufficient material for subsequent experiments, so that 80% gold recovery as a "gravity" concentrate of suitable weight proportion, *e.g.*, 10%, could not be expected by such rough concentration. The stibnite content of 17% in the original sample is considerably higher than would obtain in practice, since hand-sorting for removal of high-grade stibnite would be adopted, so that 40% stibnite is a reasonable representation of the antimony content of "gravity" concentrate.

A portion of this "concentrate" crushed to pass a 40 mesh sieve floated in slightly acid water containing a little wood tar (about $\frac{1}{2}$ % on the weight of ore treated) yielded:—

Froth (40% by weight): Au, 35 dwt. per ton; stibnite, 89%.

Tailing (60% by weight): Au, 42 dwt. per ton; stibnite, 6%.

Approximately, therefore, 40% of the gold was in the flotation concentrate, and 60% in the flotation tailing. The float also contained a little copper (0.8%) which, as will be seen later, has an important bearing on the gold extraction.

Amalgamation of the flotation tailing by grinding in a mortar gave a residue of 20 dwt. per ton, which may be taken as equivalent to an extraction of 50% on the tailing gold, or 30% of the total concentrate gold.

The gold actually recovered from the mercury represented an extraction of 24 dwt. per ton from the 42 dwt. tailing, but was probably somewhat impure.

Cyaniding of the Amalgamation Residue.—Leaching for three days gave a final residue of 9.6 dwt. per ton, so that 25% of the gold in the flotation tailing, or 15% of the original concentrate gold, was dissolved by cyanide.

Treatment of the Sand Tailing from "Gravity" Concentration.—Gold value 4.6 dwt. per ton. Leaching for three days with 0.1% KCN gave only 0.3 dwt. per ton dissolved.

Another portion of this sand, recrushed to pass a 40 mesh sieve, was passed over the Wilfley table, and the small quantity of concentrate obtained (4% by weight) on amalgamation gave a recovery of 25% of the gold in the sand.

(4) Weathered ore (Au 56.2 dwt. per ton) containing antimony as oxide and sulphide, also a little copper as sulphide and carbonate; 2,000 grams of this sample, crushed to pass a 20 mesh sieve, was passed once over the Wilfley table, yielding:—

Concentrate (and middling) 30% by weight: Au 135 dwt. per ton = 73% of the gold.

Sand tailing, 60% by weight: Au 20.9 dwt. per ton.

Slime tailing (by diff.), 10% by weight: Au 20.7 dwt. per ton.

Flotation of the concentrate, recrushed to pass a 60 mesh sieve:—

Froth, 12% by weight (Au 131 dwt. per ton).

Tailing, 78% by weight (Au 129 dwt. per ton).

Copper carbonate was dissolved during flotation, and on precipitation on aluminium gave 1% Cu on the concentrate.

Amalgamation of the flotation tailing gave a gold recovery of 60% on the tailing itself, or 47% on the concentrate, or 34% on the original ore.

Cyaniding of the amalgamation residue.—Leaching for three days dissolved gold only to the extent of 1.6 dwt. per ton on the amalgamation residue of 53 dwt. per ton, the cyanide solution having to be renewed several times.

The following observations may be made on these rather unsatisfactory results:—

(i) The removal of stibnite by flotation does not give a tailing from which a high gold extraction can be readily obtained, and the assumption that the stibnite is the chief

difficulty may be largely incorrect; for this reason further direct treatment tests were desirable under conditions not previously tried, *e.g.*, finer grinding.

(ii) The sulphide froth, in acid flotation of comparatively coarse ore, contains gold per ton roughly equal to the gold content portion of ore treated.

(iii) The presence of copper in small but regularly occurring quantities indicated a possible cause of poor extractions.

(iv) Amalgamation results are reasonably good, and the small amounts of sulphide minerals left in the tailing did not seriously affect the mercury.

Before proceeding to investigate direct treatment some flotation tests were required by the method usually adopted in practice, *viz.*, agitation with a suitable oil or oils. One of the usual laboratory flotation machines, consisting of an agitation chamber communicating with spitzkasten where the froth overflows, was used. Recent improvements, such as introduction of air at the bottom of the agitation chamber were not available, and no exhaustive tests with different oils and "reagents" were made; it must, therefore, be understood that the specialists in this method, which is a patented process, would probably obtain much better results. The figures obtained in these trials are summarised below.

Direct Flotation of Dry Ground Ore.

(5) Sample consisted of fresh stibnite ore (Au 11.2 dwt. per ton; stibnite, 15%) dry crushed to pass a 90 mesh sieve. About $\frac{1}{2}\%$ on the weight of ore of wood tar was added to the water before adding the ore. The proportion of water to ore was 4:1.

Products: Froth, 20% by weight;

Au, 9.4 dwt. per ton.

Tail, 80% by weight; Au 11.3 dwt. per ton.

In a similar test using coal tar and a little creosote a rather better froth was obtained, but as in both tests there was obviously a very considerable quantity of sulphide left in the tailing, no extraction tests were attempted. It may be noted, however, that the gold distribution is about the same as in the acid flotation tests; that is, the gold content of the float is approximately the same per ton as that of the original ore treated.

It is well known that flotation tests should be done on wet crushed ore whenever possible, the explanation being that the films of oxide formed on the sulphide particles during dry crushing have a serious

effect on the flotation. Facilities for wet crushing of small samples (an Abbe pebble mill) were kindly provided by Professor Stanley in the Metallurgical Department of the Witwatersrand University, and a flotation test on similar ore, using coal tar and creosote, gave the following figures:—

Direct Flotation after Wet Grinding.

(6) The original sample of ore (Au 14.8 dwt. per ton) broken to pass a 10 mesh sieve then ground in the pebble mill with coal tar and creosote, on flotation, gave:—

Froth, 26% by weight (Au, 20.9 dwt. per ton; stibnite, 42%).

Tailing, 74% (by diff.) Au (calculated) 12.5 dwts per ton; stibnite, 8%).

The tailing without drying was agitated with KCN of .09% strength for three hours, after which treatment the residue assayed 6.3 dwt. per ton. Although the residue was still high, there are one or two noteworthy features about this test. First, the improvement in flotation caused by wet crushing; second, the greater tendency for the gold to go with the froth on finer crushing (95% of the pulp for flotation passed a 90 mesh sieve, and 80% passed a 150 mesh sieve); third, that the cyanide was not destroyed to the same extent as in tests on dry crushed or dried pulp; in this test only 50% of the cyanide was destroyed, while on dried samples the cyanide in the first solution applied is generally completely consumed. In this, as in all cyanide tests, the alkalinity was kept very low so as to prevent antimony dissolving in the solution—notwithstanding this all the solutions after treatment contained antimony.

The results so far obtained indicated that a considerable proportion of the gold was very finely distributed and probably "encased," so as to remain unexposed to the action of either cyanide or mercury, and that this, rather than the presence of stibnite, might be the cause of low extractions. The following comparative tests were therefore done on a sample of fresh sulphide ore, containing gold 9 dwt. per ton, and stibnite 8.5%. This may be regarded as a rather typical sample of the unweathered ore of the district.

The total sample was coarsely broken to pass a 10 mesh sieve.

Direct Amalgamation and Cyaniding.

(7) Sample (800 grams) ground wet with mercury for two hours (90% of the ground

pulp was — 150). The gold recovered from the mercury was equivalent to an extraction of 70%, and the residue assay of 2.5 dwt. per ton is equivalent to 72% extraction by direct amalgamation in presence of the stibnite. A small quantity of concentrate was separated while panning off the mercury, which assayed only 10.4 dwt.

Cyaniding of the Amalgamation Residue.—With air agitation this gave the surprisingly low residue value of 1.2 dwt. per ton, and the cyanide strength which originally was 0.11 KCN, was .06% KCN at the end of the test. Direct amalgamation and cyaniding tests done previously on dry ground ore had usually not yielded more than 40% to 50% of the gold, while in this case wet grinding and amalgamation, followed by cyaniding of the amalgamation tailing, gave 87%. The improvement must be due to the more efficient "sliming" effect of wet grinding, and it is also possible that the presence of plenty of water reduces the tendency to "flouring" of the mercury. It had been found in an earlier test that when the proportion of water added was only sufficient to make a paste, "flouring" was much worse than when the proportion of water was sufficient to keep the pulp in the form of a thin cream.

Flotation followed by Amalgamation and Cyaniding of the Tailing.

(8) For purposes of comparison a portion (500 grams) was wet ground in the same way for four hours with $\frac{1}{2}$ % coal tar and creosote, so that the whole was reduced to a slime (—200) and the finely ground pulp on flotation yielded—

Froth (29% by weight), Au 19.8 dwt. per ton; stibnite, 28%.

Tailing (71% by weight), Au 4.2 dwt. per ton; stibnite, 1.3%.

i.e., approximately 65% of the gold was in the flotation concentrate and 35% in the tailing. This is a very different flotation result as regards gold distribution than the earlier tests, when much coarser material was treated, and indicates either that minute particles of gold are set free from "gangue" by the finer grinding, or that there is some other floatable mineral present with which gold is closely associated and which requires very fine grinding to separate it sufficiently for flotation. With such finely ground ore the flotation concentrate contains a much larger proportion of gangue particles, and were the antimony the main consideration the result could hardly be called good. Portions of the flotation concentrate and

tailing were cyanided in the air agitator with the following results:—

Flotation Concentrate: Au 19.8 dwt per ton.

Cyanide Residue: 16.8 dwt. per ton.
= 10% extraction on the original sample.

Flotation Tailing: Original value 4.2 dwt. Cyanide residue, 1.5 dwt.

= 64% extraction on the tailing, and 23% on the original sample.

Comparing tests Nos. 7 and 8 there is obviously nothing to be gained by initial separation of the stibnite by flotation.

In order to confirm if possible these greatly improved cyanide extractions due to fine grinding, a series of four short cyaniding tests was carried out on three portions of fresh sulphide ore.

(9) Au 7.1 dwt. per ton; stibnite, 10%; grading of sample—40% + 90; 30% + 200; 30%—200.

Residue after $1\frac{1}{2}$ hours cyaniding, 5.4 dwt. per ton.

(10) Same sample as above, but ground so that 95% passed a 200 mesh sieve.

Residue after $1\frac{1}{2}$ hours, 4.2 dwt. per ton.

(11) Sample unfortunately not quite the same; contained only 8% stibnite and Au 9.0 dwt. per ton; wet ground to—200.

After $1\frac{1}{2}$ hours cyaniding the residue was 3.1 dwt. per ton.

(12) Sample same as No. 11, but the cyanide solution used was much weaker (.01%). Mr. J. M. Thorburn had suggested to the writer that there might be a disadvantage in using such "strong" solution as .09% KCN, since the action on the stibnite would be greater, and in any case the weaker solution would be just as effective in dissolving gold. The cyanide in the first solution applied was all destroyed. The second solution was .003% at the end of the test; but after three hours agitation the residue was 4.4 dwt. per ton.

These figures are hardly as confirmatory as could be desired; probably the cyaniding period was not continued long enough. They point again, however, to the necessity for wet grinding to a very fine product. In this test further confirmation of the bad effects on the cyanide of dry grinding or artificial drying of samples to be cyanided was obtained. In tests Nos. 9 and 10 the cyanide in the first solution applied was completely destroyed, while in test No. 11 (wet grinding) the consumption of cyanide was as shown.

Original strength, 0.9% KCN.

After $\frac{1}{2}$ hour, .05% KCN.

After 1 hour, .05% KCN.

After 1½ hours, .045% KCN.

The solutions were all faintly alkaline at the end of the tests.

From the foregoing rather miscellaneous collections of tests, the following points are clear:—

(i) Even with considerably improved flotation results the gold will not be sufficiently isolated in either flotation concentrate or tailing for one or other to be rejected.

(ii) Direct amalgamation on wet ground ore, when the antimony content is not unduly high, is quite satisfactory (the method of amalgamation used, however, probably gives much longer contact than would be possible in practice).

(iii) Although better results are obtained by fine grinding, cyaniding is, on the whole, unsatisfactory, since in addition to poor extraction, the consumption of cyanide is high and the gold bearing solution would certainly give serious precipitation difficulties.

In view of these points the application of flotation to these ores would simply be in place of the usual cyaniding, and as a superior method of concentration to the ordinary gravity methods.

In order to test this procedure, a sample of mixed ore such as might represent the output of a mine was treated as follows:—

Amalgamation followed by Flotation.

(13) 600 grams of the sample (Au 20.5 dwts. per ton, and sulphur as sulphide 3.2%) was dry crushed to 20 mesh, then wet ground in the pebble mill with mercury for one hour. After panning off the mercury the tailing was re-ground with the addition of ½% (coal tar and creosote) till about 90% passed 150 mesh sieve and the pulp floated yielding—

Froth, 20% by weight, Au 33 dwt. per ton (S. 12.6%).

Tailing, 80% by weight, Au 3.6 dwt. per ton (S. 0.9%).

The above is equivalent to an 88% gold recovery, i.e., 55% as amalgam and 33% as flotation concentrate, which can be considered as fairly satisfactory as a first test. Further investigation of the best conditions for flotation would probably give improved results. From the practical point of view the weakness of the above test (*No. 13*) is that the whole of the sample was ground with mercury, and a further test was done in which only a "gravity" concentrate was amalgamated.

Amalgamation, "Gravity" Concentration and Flotation.

(14) 2,000 grams of a sample very similar to that used for No. 13, Au 21.0 dwt. per ton, crushed to pass a 20 mesh sieve, was passed once over the Wilfley table, and yielded:—

Concentrate (25% by weight), Au 48.8 dwt. per ton (roughly 60% of the gold).

Tailing (75% by weight), Au 10.0 dwt. per ton (roughly 40% of the gold).

Amalgamation of the concentrate (ground wet with mercury for three-quarters of an hour): Residue, 29.0 dwt. per ton (Au recovered, 15 mgms.).

The tailing was divided into two portions, one for a flotation test and the other for cyaniding.

Flotation, sample ground with ½% coal tar and a little pine oil added during flotation:—

Froth (20% by weight), Au 36.5 dwt. per ton.

Tailing (80% by weight), Au 2.2 dwt. per ton.

Summarising these figures—

24% of the gold was recovered as amalgam.

34% of the gold was in "gravity" concentrate.

35% of the gold was in the flotation concentrate.

Cyaniding.—A comparative cyaniding test on the remaining portion of tailing was kindly carried out by Mr. Thorburn after re-grinding to the following grading:—

+ 60, 0.3%; + 90, 6.3%; + 200, 26.8%; — 200, 66.5%.

This test was carefully conducted with .008% KCN solution, but the residue after 30 hours' treatment was 8.48 dwt. per ton, which amply confirmed the view that cyaniding was useless, being a worse result than those obtained by the writer with much stronger solutions.

A treatment scheme of amalgamation and concentration involves facilities for roasting and extracting the gold from the flotation concentrate. The best conditions for roasting such a concentrate require some investigation, particularly if as much of the antimony as possible is to be recovered as the valuable white oxide. The writer's experience is that the best roast for gold extraction is at a very low temperature with the maximum quantity of air, which conditions are unfortunately not suitable for the production of

the white oxide, Sb_2O_3 . Probably a compromise, consisting of alternating reducing and oxidising conditions would be found feasible. As an example of the extraction obtained on roasted ore the following figures, obtained some years ago, may be given:—

Roasting at Very Low Temperature with Excess Air.

(15) Sample contained about 10% stibnite and Au 12 dwt. per ton. Leaching the roasted product for three days with cyanide dissolved 80% of the gold.

Example of Volatilisation Roast.

(16) The sample contained 24% of antimony as stibnite and was mixed with 10% of coke dust. It was roasted at a dull red, and the residue, which was in a semi-fused state, contained 4.3% antimony and gold 13 dwt. per ton; 50% of this gold was recovered by grinding with mercury.

The copper in these antimonial ores has been remarked on more than once, and it is well known in the district that its presence is an indication of high gold values. It occurs very regularly, but seldom in any quantity above 1%. The mineral in which it occurs is not readily attacked by HCl , so that in decomposing samples for antimony estimations, the copper is left with the "silica and insoluble" portion (this, of course, refers to unoxidised sulphide ore). It was suspected that its presence might have some bearing on the low cyanide extractions, and microscopic and chemical examination of concentrate confirmed this to some extent. The unweathered sulphide is a dark-grey black mineral, which oxidises in the air much more readily than stibnite yielding copper carbonate and oxide of antimony. An analysis of this mineral separated from the associated stibnite by treatment with HCl (stibnite is easily decomposed by HCl , but the copper bearing mineral is hardly attacked), gave Cu 41.2%, Sb 23.7%, S 21.6%, the remainder being "insoluble," and a little iron. This is, roughly, the composition of the mineral Tetrahedrite ($\text{Cu}_8\text{Sb}_2\text{S}_7$).¹ A few fragments of the mineral picked out with the forceps gave, on scorification, gold to the value of 2 ozs per ton. Twenty grams of the flotation concentrate obtained in No. 3, which, according to the assay, should have

contained 1.2 mgms. Au, were treated with HCl to remove stibnite, then with a little mercury to remove any free gold; the residue was then separated from the small proportion of silicious gangue and scorified, giving 1.1 mgm. of gold. This appears to be a case somewhat similar to that mentioned by Green,¹ that is, minute quartz particles with attached gold more or less enclosed by sulphide; even the apparent clean mineral fragments leave a trace of silicious residue on decomposing with nitric acid. White, in discussing Green's results, states² what probably applies equally to these antimonial ores: "... behind all apparent difficulties connected with the associated minerals the real obstacle to a high extraction lies in the exceeding smallness of a large portion of the gold particles." In samples from the Murchison range a few specks of coarse gold can sometimes be seen, but none were observed in any of the samples dealt with in this investigation, and microscopic examination of concentrate washed down to a very small bulk failed to show gold with certainty. Treatment of rich concentrate particles on a slide with HNO_3 , however, revealed small particles of gold very distinctly.

In concluding these notes a few general remarks on the Murchison Range district, based on some personal knowledge, may not be out of place. It is obviously a mining field which at present is regarded as too speculative for the larger mining companies, but for the "small worker" or tributary syndicates, it would be very suitable, provided the great objection of difficult ore treatment were surmounted. It was partly in the hope of making suggestions for a simple treatment scheme, such as could be undertaken by reasonably experienced men, that this investigation was carried out. Success would undoubtedly depend largely on the establishment of a suitably located works where concentrate, obtained either by flotation or simple "gravity" methods, could be treated.

If this were done there is no reason why ore of reasonably high gold content should not be profitably mined. Flotation following amalgamation would be required for the best recovery, and there is no particular reason why flotation should not be

¹ A. L. Hall, Memoir, No. 6, S.A. Geological Survey, mentions the occurrence of "grey copper ore."

¹ *Jour. of Chem., Met. and Min. Society*, September, 1912, p. 98.

² *Ibid.*, p. 101.

used on small mines. In any case, the installation of a flotation plant can always be postponed till returns are being obtained by simple amalgamation and "gravity" concentration. In order to get the best amalgamation results from a simple plant it would be necessary to concentrate as closely as possible; after running the ore over plates and re-grinding this concentrate with mercury, the residue from this second amalgamation could then be sent to the assumed smelting works along with flotation concentrate, if that were produced.

The problem of the treatment of these ores is worthy of considerable investigation because of the great extent of the deposits, but it is useless for people to imagine that ore of 6 or 7 dwt. per ton could be profitably treated; the payable minimum would probably be about twice this figure. It is also useless to count on any considerable revenue from the export of antimony ore; the best thing to do with the antimony would be to recover it as oxide for use as paint.

Thanks have to be recorded to Professor Stanley and Mr. Millin of the Witwatersrand University, for providing testing facilities in the Metallurgical Department of the University; and to Mr. J. M. Thorburn, for valuable assistance in assaying.

OBITUARY.

HANS SOREN MEYER.

Hans Soren Meyer, who so suddenly passed away on the 3rd of July, was born in London in 1878, of Danish parents. His father was prominent in London diplomatic circles, and the son began life under favourable auspices. He took up the study of mining engineering at the Royal School of Mines, and after graduation, carried on his thorough and diversified training at silver mines in Norway and coal mines at Wales.

Mr. Meyer began his Witwatersrand career in 1898, being engaged in the various capacities of sampler, smelter and surveyor on such well-known early mines as the Crown Reef, Robinson and Geldenhuis. He joined the Consulting Engineers' Staff of the Werner-Beit group in 1902, where he rendered conspicuous service for twenty years, up to the time of his death.

With a quiet personality, Hans Meyer joined great firmness and perseverance. He was a good and loyal friend, and possessed an inexhaustible reservoir of humour which lighted up many a perplexing situation. Professionally, he was well known as an engineer of sound knowledge and judgment; incisive and fearless in expression.

The activities and welfare of the Society were of great interest to him. He became a member in July, 1899, and served on the Council from 1912 up to the time of his decease. He was Vice-President during the year 1917-1918 and President in 1918-1919.

Though not a frequent contributor to the *Journal*, he regularly attended the meetings of the Society, and his participation in the various discussions was always marked by shrewd criticism and enlightening observations.

The deepest sympathy is extended to the widow in her bereavement.

F.W.

Notices and Abstracts of Articles and Papers.

CHEMISTRY

CHEMISTRY OF PLATINUM AT HIGH TEMPERATURES AND PRESSURES.—In an article published in the December number of *Economic Geology*, Mr. C. R. Shaw describes researches made by him with the object of throwing light on the origin and formation of platinum in ore deposits. His results and summary are as follows:—

"The marked solubility of platinum in solutions of ferric and cupric chlorides renders the theory of platinum transportation by chloride waters entirely plausible. When heat and pressure are applied by hydrochloric acid alone or even the alkali chlorides in the presence of an oxidizing agent a significant solvent action on the metal takes place.

"A study of the decomposition of hydrochloroplatinic acid under the influence of temperature and pressure shows it to be a most unlikely medium of transportation, decomposition being complete at 175° in twelve hours. Decomposition proceeds at much lower temperatures, but more slowly. The salts of this acid, the chloroplatinates, however, are much more stable compounds, and are capable of existing up to about the critical temperature of water, 370°. They form, presumably, the media by which platinum is transported in chloride solution.

"Hydrated platonic oxide, so easily formed by contact of the chloroplatinate solutions with carbonate rocks, is a very stable compound. Decomposition of the pure material is only apparent in twelve hours at 390°. The rate of

decomposition at considerably higher temperatures is rather slow. The presence of the chlorides of sodium and calcium has little effect upon the decomposition. Magnesium chloride, however, exerts a very marked effect in lowering the point of decomposition. Morris observed a similar effect in his study of auriferous hydroxide. Cribb has shown that magnesium chloride solutions, when heated in a sealed tube, are hydrolyzed with the formation of the oxide and hydrochloric acid. In the experimental work on platinum oxide and magnesium chloride this hydrochloric acid would dissolve the platinum oxide, forming hydrochloroplatinic acid. This is decomposed at lower temperatures than is the hydrated oxide. In this way the action of magnesium chloride can be explained. The action of manganese chloride was to appreciably decrease the rate of decomposition.

"When various carbonates were added to hydrochloroplatinic acid and heated under pressure, impure hydrated platinum oxide was first precipitated upon the crystal. The action of the heat and pressure then decomposed the oxide, metal being deposited upon the crystal as a coating. In this way some excellently plated specimens were obtained. In agreement with preceding experiments, magnesite and siderite required lower temperature for reduction, while rhodochrosite required higher temperature than did calcite. The effect of contact of chloroplatinic solutions with carbonate or basic rock is at once evident. The hydrated oxide would be formed, to be subsequently reduced by some agency such as heat and pressure or reducing solutions."

SUMMARY.

A discussion of the formation of secondary platinum deposits is presented.

Platinum is readily soluble in various chloride solutions, especially when heat and pressure are applied. Ferric chloride and cupric chloride dissolve platinum at atmospheric pressure, but much more rapidly at 160°. Sodium chloride and manganese dioxide, and even hydrochloric acid, are solvents at that temperature.

Reduction of hydrated platinum oxide to metal under the pressure of aqueous vapour is perceptible in twelve hours at 390°. At 525° reduction is complete. The reduction proceeds in two steps, the lower oxide, PtO, being first formed.

Sodium chloride and calcium chloride do not appreciably affect the temperature or speed of reduction of hydrated platinum oxide. With magnesium chloride, however, reduction to metal takes place at a temperature 80° to 90° below that necessary for the reduction of pure oxide. Manganese chloride decreases the rate of decomposition very appreciably.

Reduction of hydrochloroplatinic acid solution is complete at 175° in twelve hours. Some reduction took place at a temperature of 147°.

The reaction of carbonates and hydrochloroplatinic acid at high temperatures and pressures takes place in two steps. A more or less impure hydrated platinum oxide is first thrown out upon the carbonate. This is then reduced by the action of heat and the pressure of aqueous vapour to metal. The temperature of reduction varies with the carbonate used, being

lowest with magnesite and siderite, and highest with rhodochrosite.

Platinum is thrown out of hydrochloroplatinic acid upon serpentine and takes the fibrous form of the serpentine.

Ferric chloride added to hydrochloroplatinic acid solution prevents any reduction to metal at a temperature over 100° above the point of complete decomposition of the acid alone.

Solutions of sodium, calcium and magnesium chloroplatinates are completely decomposed at a temperature of 370° in twelve hours, about that of the critical temperature of water.—G. R. SHAW, *Economic Geology*, Vol. XVI., No. 8, December, 1921, p. 524. (F.W.).

THE DETECTION AND ESTIMATION OF PLATINUM IN ORES.—Methods for the determination of platinum in ores may be separated into two classes. The first includes those known as "wet" methods of analysis. These are used on concentrates and material containing considerable quantities of the platinum metals, and are not applicable to the estimation of platinum in low grade ores. The second class—fire-assay methods—are the only ones available at the present time for the determination of small amounts of platinum in vein material. Although claims have been made that ore has been found in which the platinum occurs in such a form as to escape detection by either the "wet" treatment or the fire assay, the author doubts the existence of such material, and an authority is quoted in the following sentence: "From all available information such reports should be regarded with suspicion, for it is confidently believed that any platinum ore of commercial grade will yield to standard methods of analysis if made by competent and reliable analysts."

F. P. Dewey, chief assayer, Bureau of the Mint, Washington, D.C., says: "Methods which would unlock platinum not usually disclosed by fire assay, I class in the same category with gold making. . . . I have also had considerable experience with these platinum fakes."

A number of assays on materials containing small pieces of platinum, finely divided precipitated metal, and two differently prepared platinum colloids were carried out, and in all tests the platinum was recovered without difficulty.

The results of these experiments, coupled with the results obtained by reputable assayers, not only shows the possibility of the recovery of colloidal platinum by the fire assay, but also indicate that excessive temperatures are not essential to the recovery of finely divided platinum by this means.

The selected method for the commercial estimation of platinum in ores is as follows:—

The ore in all cases is ground to pass 100-mesh screen, and when refractory substances such as chromite and zircon are present, the sample should be made to pass a 150-mesh screen. When the quantity of platinum is small it is advisable to run several samples of an assay ton each, combine the lead buttons, and scorify to about 30 grams.

An assay ton of the carefully sampled ore is thoroughly mixed with a suitable flux. As iron has a greater affinity for iridium than has lead

the old iron-nail method of treating sulphide ores should not be used.

If there is not at least fifteen times as much silver as platinum in the ore, enough of a silver salt is added to make up the deficiency. This is required to make the platinum soluble in nitric acid when parting, to remove last traces of lead in cupelling, and to lessen any cupel loss.

To an unknown ore about 0.05 gm. of silver chloride may be added. The charge is fluxed as in the ordinary gold and silver fire assay, except that after the fusion is quiet the temperature should be raised somewhat higher than usual, and the heating continued for about an hour. The crucible is removed from the furnace without agitating the contents, and left to cool.

The button is freed from slag and cupelled at a high temperature. As the high temperature causes a loss of silver, a check is run to correct this error.

When platinum makes up over 1.6 per cent. of the bead, the latter appears frosted to the unaided eye. The microscope will detect platinum when it constitutes 0.3 per cent. of the silver bead. Considerable quantities of platinum make the bead flat and irregular. The roughness caused by iridium is of finer texture than that due to platinum. Palladium gives the surface of the bead an embossed appearance. Ruthenium turns the surface to a bluish-black colour, a black scum being left on the cupel.

After cupelling, the button is parted with nitric acid, first with 1:4, then with 1:1, and finally with 2:1 acid.

If gold, silver and all the platinum metals are present in the ore, the silver, palladium and platinum are dissolved, leaving the gold, iridium, rhodium and some ruthenium and osmium. Part of the iridium may not collect in the silver and will be lost on the cupel.

If considerable platinum is found, some will be left still undissolved. The residue is filtered off, ignited, and saved for the recovery of residual platinum and for determination of iridium and rhodium. Dilute hydrochloric acid is added to the filtrate slowly, with constant stirring to precipitate the silver. After standing overnight the silver chloride is filtered off; if it is pink it must be redissolved and reprecipitated to remove occluded platinum or palladium.

Evaporate the filtrate just to dryness, do not bake; take up with dilute HCl and again evaporate just to dryness; repeat this process, and after cooling take up with a very little cold water and filter off any traces of silver. Make slightly alkaline with sodium carbonate, add formic acid, and boil thirty minutes to precipitate platinum and palladium. Filter off these metals, ignite and weigh. Test the filtrate for complete precipitation by boiling with more formic acid. If the first parting solution is yellow or orange, the presence of palladium is indicated, and platinum and palladium may be separated by warming the metals with dilute nitric acid 1:4, which dissolves palladium. The insoluble platinum is weighed and the palladium determined by the difference.

The original insoluble residue from the first parting acids may contain gold, iridium, rhodium and some ruthenium, osmium and un-

dissolved platinum. It is filtered off, ignited and weighed. Treatment with dilute aqua regia dissolves gold and platinum, and the insoluble metals are weighed together.

The gold and platinum solution is evaporated several times to dryness with hydrochloric acid, filtered if necessary, and the gold precipitated by boiling with oxalic acid. After standing overnight the gold is filtered off, and platinum precipitated from the neutralised filtrate with formic acid as above, its weight being added to that of the platinum already found.

Checks containing known amounts of gold, silver and platinum are done, and platinum and palladium positively identified in the sample by the usual potassium iodide tests, etc.

An optional colorimetric method of estimating small quantities of platinum is described in detail in the original paper, and various reactions of platinum are given.

Methods of determination of platinum used by different assayers are described and criticised. Schoeller and Powell's methods for the detection and determination of the platinum group metals are tabulated.

The author states that a method often used, but not to be recommended, is known as the "difference" method. The bead obtained by the regular fire assay is parted with sulphuric acid, which dissolves the silver and leaves gold and platinum; the latter metals being annealed and weighed together. Silver is then inquarted and the bead parted with nitric acid, dissolving silver and platinum. The residual gold is annealed, and the platinum found by difference. This procedure is to be avoided for the following reasons:—

(1) It makes no attempt to group the platinum metals in a way to assist the evaluation of an ore.

(2) Parting with sulphuric acid is unsatisfactory. Not only does boiling sulphuric acid attack platinum, as shown by McCay and by Steinmann, but the procedure is disagreeable.

Although slightly dilute sulphuric acid at a definite temperature may be used, the action of the several platinum metals in this treatment is uncertain.

Methods that determine platinum by "difference" have a twofold chance of error, that due to parting with sulphuric acid, and that due to the fact that platinum may not be entirely dissolved by one treatment of the gold-platinum bead (inquarting silver and parting with nitric acid).—C. W. DAVIS, *Technical Paper*, 270, U.S. Bureau of Mines, 1921. (R.A.C.).

DIRECT TITRATION OF ALKALINE MANGANATES.—Dr. Bloch stated that all published methods for the titration of alkaline manganates or permanganates necessitated the acidification of the same previous to the titration, or during titration. There were obvious disadvantages to such methods, especially when a prompt determination was required.

The method described in the note was not essentially new, being based on the known reaction:—

$$2 (K Mn O_4) + 3 (Mn SO_4) + 2 Zn O = 5 Mn O_2 + K_2 SO_4 + 2 Zn SO_4.$$

Similarly it had been possible to titrate an alkaline permanganate according to the reaction.

$2 \text{ Na Mn O}_4 + 3 \text{ Mn SO}_4 + 4 \text{ Na OH} = 5 \text{ Mn O}_2 + 3 \text{ Na}_2 \text{ SO}_4 + 2 \text{ H}_2 \text{ O}$, and an alkaline manganate as follows:—

$\text{Na}_2 \text{ Mn O}_4 + \text{Mn SO}_4 = \text{Na}_2 \text{ SO}_4 + 2 \text{ Mn O}_2$.

In practice the titration was done at 60–70° C., the manganate or permanganate solution being diluted with hot water to 250–300 c.c., and a N/2 solution of manganese sulphate was run in slowly, a precipitate of hydrated manganese dioxide being formed, and it was allowed to settle, and the addition continued until the supernating liquor was colourless.

The N/2 manganese sulphate solution was prepared from pure manganese sulphate free from iron, and the solution standardised against a normal standard permanganate solution in the presence of zinc oxide.

After titration of a sample, an aliquot portion of the clear decanted solution may be titrated with N/10 sulphuric acid to ascertain the excess of alkali over the quantity required by the given equations.—Dr. J. J. Bloch. *The Chemical Age*, December 10, 1921, p. 731. (R.A.C.).

TWO FORMS OF ISOTOPES.—In a communication of the *Chemical News*, of October 28th last, Professor Bohuslav Brauner, of Prague, presents in some detail a study of the question of isotopy in connection with the international revision of atomic weights. Some chemists have expressed the view that the exact determination of these weights, and the statement of them to several places of decimals, have lost a great deal of importance. The atomic weights of an element may represent only that of the average mixture of its isotopes. Professor Brauner points out that at present two classes of isotopes are known. One class consists of radioactive elements, products of the same or different series of radioactive disintegration (Soddy's isotopes), the other class (isotopes of J. J. Thomson and Aston) represents mixtures of extremely similar elements, which were formerly regarded as individual elements respectively. To this class belong the chlorine isotopes obtained by Harkins.

The first class of isotopes descends from uranium or thorium, and the different series end in "leads" of the atomic weights 306 and 208, but all such representing one place do not necessarily occur together. Those of the second class, so far as now known, always occur in association, in the same proportion, the mixture always having the same average combining weight. Aston has shown that separation can be affected by electromagnetic analysis, and that their atomic masses are represented by integers on the base 0=16. They run to-day from hydrogen to caesium. Incidental to these considerations, a modification of Mendeléef's law is necessary in regard to the isotopes, but it may still stand with reference to the pure elements. As a conclusion from his studies Brauner recommends that the term "atomic masses" be used only for the whole numbers of the isotopes of the second class, while the term "atomic weights" should be applied to the numbers obtained by chemical methods and the use of the balance, no distinction being made between pure elements, mixtures of isotopes or

single isotopes.—B. BRACNER, *Journal of the Franklin Institute*, January, 1922, p. 116. (J.A.W.).

DETERMINATION OF ARSENIC.—Advantage is taken of the intensifying and fixing action of potassium iodide on the yellow stain produced by hydrogen arsenide on mercuric chloride paper to provide a new method for the quantitative estimation of small quantities of arsenic compounds. The apparatus required consists of a flask fitted with a vertical glass tube about 5 mm. in diameter and 30 cm. long. The lower end is drawn out into a dropping tube, and a hole in the tube about 2.5 cm. from the end allows of the exit of the gas. For each determination the flask is charged with 8 g. of zinc and 60 c.c. of 20 per cent. sulphuric acid, and the arsenic solution, previously oxidised by means of a few drops of permanganate in order to convert sulphur or phosphorus compounds into non-reducible sulphuric and phosphoric acids, is added all at once. The lower part of the glass tube contains a roll of filter paper about 10 cm. long, and then a band of the mercuric chloride paper 5 mm. wide and 12–15 cm. long, extending to within 2 cm. of the roll of filter paper, which protects it from moisture, carried up mechanically by the gas. At the end of the experiment the mercuric chloride paper is immersed in 10 per cent. potassium iodide solution which acts specifically on the yellow arsenic stain, and turns it brown, and at the same time renders it permanent to light and moisture. The arsenic is then estimated by comparing the stain with those produced under similar conditions by known quantities of arsenic.—J. CRIBIER, *J. Pharm. Chim., Journal of Society of Chemical Industry*, November 15, 1921, p. 792a. (A.W.).

DETERMINATION OF POTASSIUM IN THE PRESENCE OF SODIUM, MAGNESIUM, SULPHATES AND PHOSPHATES.—The mixture of salts (*e.g.*, KCl, 0.5, Na₂SO₄, 10H₂O, 1, MgSO₄, 7 H₂O, 0.6, and NaH₂PO₄, 1.1 g.) is twice evaporated with perchloric acid until fumes of the acid are evolved, the residue is treated with 100 c.c. of methyl alcohol and the mixture boiled for one hour; after a further eighteen hours the potassium perchlorate is collected on a filter, washed with 50 c.c. of methyl alcohol containing 5 per cent. by vol. of perchloric acid (sp. gr. 1.12), then with 20 c.c. of ethyl alcohol, dried and weighed.—H. ATKINSON, Analyst, *Journal of Society Chemical Industry*, October 31, 1921, p. 751a. (A.W.).

BLEACHING POWDER IN THE TROPICS.—Many attempts have been made to render bleaching powder stable at high temperatures. Rettie, Lorrain-Smith, and Ritchie (J., 1918, 311r) have investigated the stability of dry bleaching powder, and found that bleach mixed with 20 per cent. of powdered quicklime ("stabilised bleach") lost its available chlorine, at high temperatures, at a very slow rate compared with ordinary bleach.

A consignment of 2 cwt. of this "stabilised bleach" was received by the Government Medical Stores Depot, Madras, in June, 1919, for a test of its stability in India. The product arrived in cases (three-ply wood) containing 1

ewt. each, and was immediately transferred to 2-lb. amber bottles with ground-glass stoppers. The powder was tested over a period of two years alongside a sample of ordinary bleaching powder received at the same time. The sample of ordinary bleaching powder arrived in a clear glass bottle inside the box of stabilised bleach and was immediately transferred to an amber-coloured glass bottle with ground-glass stopper. The following were the results obtained:—

	Ordinary Stabilised bleach. bleach. % Cl. % Cl.	
Analysis in England ...	35	25
On arrival in Madras ...	20.04	18.4
6 weeks after arrival ...	12.52	16.56
10 " " " " "	4.68	15.9
14 " " " " "	9.42	15.05
26 " " " " "	—	12.08
42 " " " " "	—	10.0
52 " " " " "	—	8.5
104 " " " " "	—	3.48

It is thus obvious that "stabilised bleach" is a much more reliable article than ordinary bleaching powder for use in the tropics. It has also the advantage that it remains a dry powder, even although it contains 3.5 per cent. of available chlorine and has been kept for two years in a high temperature. The rate at which bleaching powder loses its available chlorine increases owing to the formation of water during decomposition. The rate of decomposition of "stabilised bleach" is fairly steady.

After two years in Madras this sample of stabilised bleach was found to contain 5.9 per cent. of calcium chlorate, so that it would not be advisable to use "stabilised bleach" for sterilising water after a period of one year, as this amount of chlorate would give a nauseating taste to the water.

The author is indebted to the Director-General, Indian Medical Service, for permission to publish this information.—CAPTAIN A. F. MACCULLOCH, M.A., B.Sc., A.I.C., Government Medical Stores Depot, Madras, India, *Journal of Society of Chemical Industry*, October 15, 1921, p. 240T. (A.W.).

DETERMINATION OF ALKALI HYDROXIDE AND circuit the Dorr type machine is common. The method for the estimation of alkali in potassium or sodium cyanide consists in titrating the cyanide with silver solution, removing the turbidity formed at the end point with a drop of cyanide solution and then titrating with N/1 acid until the turbidity reappears. The method is accurate if most of the alkali is present as hydroxide and is therefore suitable for control tests in cyanide manufacture. In presence of other weak acids, e.g., formic acid, this method cannot be used, neither does Clennell's method (J., 1903, 762) give good results. The procedure is as follows. The cyanide is first titrated with silver solution, and the same quantity of solution after filtration can then be used for alkalimetric determination. If great accuracy is not required the solution need not be filtered. If ferrocyanide is present the solution may be titrated directly with N/1 acid and methyl orange. The cyanide having been previously determined, the alkali is obtained by difference.—F. MÜHLERT, Z. angew. Chem., *Journal of Society of Chemical Industry*, October 15, 1921, p. 693A. (A.W.).

PRESERVATION OF WOOD PULPS.—Further attempts to find an efficient preservative for mechanical wood pulp in storage confirm the statement that sodium fluoride is the most satisfactory chemical for this purpose (cf. J., 1921, 577A). Zinc chloride and mercuric chloride inhibited the decay which results in the disintegration of wood pulp, but seemed to stimulate a peculiar fungus and mould growth, which caused bad discoloration of the pulp. Sodium fluoride applied in approximately 1 per cent. solution and in such quantity as would correspond to 0.581 per cent. of chemical on the air-dry pulp, gave excellent results, the pulp remaining intact after storage for eighteen months under conditions favourable to decay. The chemical was applied to the wet sheet as it was being made into laps, and was retained to the extent of about 80 per cent. of the amount used.—J. S. BATES, Pulp and Paper Mag., *Journal of Society of Chemical Industry*, October 15, 1921, p. 689A. (A.W.).

METALLURGY

THE DESIGN OF FLOTATION PLANTS.—The author reviews the modern trend in concentrating mill design, particularly in regard to flotation plants. A number of flow sheets are shown, also a detailed tabulation of features, which for lack of space are not reproduced in this abstract. Eight modern plants in Western America are analysed.

Outstanding development in design are:—(1) Simplicity of treatment scheme and compactness of plant; (2) increased use of reinforced concrete; (3) more efficient use of electric power and devices of control; (4) the tendency to use fool-proof equipment to obtain continuity of operation. Ball mills are generally operated as a complete interchangeable unit, and are picked up by cranes and replaced by spare units in a few minutes, when repairs are needed. Reinforced concrete is used in floors, elevator housings, classifier boxes, feed pipes, pounders, flotation cells, thickener tanks, excavator tanks; this common application being not only due to advantage from a construction point of view, but also due to greater finality in design and process.

For grinding, opinion is divided between the relative merits of ball mills, rod mills, and conical mills. Step comminution is generally recognised as most efficient, especially in large plants. The author advocates a return to rolls as a step in stage breaking and grinding. Disc crushers are largely used for intermediate crushing. For classification in the grinding circuit the Dorr type machine is common. The Akins screw type is also used.

For gravity concentration the use of tables continues to be common practice. Of flotation machines a large number of air cells are used as distinguished from cells of the agitator class. Smaller plants having less elaborate plant frequently prefer the latter class cell machine. The new Callow is thirty-six feet long. The new Minerals Separation machine has no separate Spitzkasten compartment for frothing, a horizontal baffle serving to restrict the violent commotion to the lower part. A number of machines in which the stirring device revolves

on a horizontal shaft have been successfully developed. Slower speed operating machines are considered to be the aim of the operators and designers of flotation machines.

A soluble flotation reagent has been introduced which diffuses rapidly and requires no agitation for this purpose.

Secondary treatment of concentrate is practised by many, in fact even where only one type of cell is used the froth from the last cells is usually returned to the head of the machine for cleaning.

Froth concentrate is being settled and de-watered almost universally in Dorr thickeners, followed by a drum type of vacuum filter. For elevating pulp, bucket elevators find preference owing to their simplicity and low power consumed.—A. B. PARSONS, *Mining and Scientific Press*, December 3, 1921, p. 775. (F.W.)

EVOLUTION OF MECHANICAL ROASTING.—The author follows the development of roasting, particularly that of lead ore, from the early days. The hand rabbled furnaces were capable of effective work, producing a thoroughly roasted product for the blast furnace. It was, however, wasteful of fuel, had a small capacity, and was extremely exhausting to the operators as well as being conducive to lead-poisoning. The physical condition of the furnace product was not well suited for the subsequent blast furnace smelting as it was too fine and granular. Attempts were made to produce what was called a sinter roast by increasing the heat at the discharge end to an incipient fusion. This produced a more suitable product, but control of this operation was difficult.

Metallurgists, particularly those engaged in roasting copper ores, gradually evolved mechanically rabbled furnaces, first the long straight line type, then the turret type, and finally the multiple hearth type, which is also self-roasting. Although these worked well on copper sulphide ores, lack of heat control prevented them from being generally adopted for lead ores.

Then briquetting came into favour, and was considered for a while to be capable of providing a solution. It was found, however, that briquettes were hard and too dense to permit penetration of furnace gases, or they disintegrated and dusted with high heat.

The next step was experimentation with blast roasting. The era of blast roasting may be said to have begun about 1902, when the Huntington-Heberlein post roasting process was made the subject of experimentation. From this time the roasting of lead ores in pots by the H. and H. process became common at large smelters. The product was superior and the efficiency per unit of grate area was 25 times that of the hand roaster.

The pot roaster was followed in 1908 by the Dwight and Lloyd roasting and sintering machine, which had a capacity per unit of grate surface 125 greater than hand roasters and 5 times greater than pots.—ARTHUR S. DWIGHT, *Mining and Metallurgy*, December, 1921, p. 5. (F.W.)

A NEW CYANIDE.—An account of German attempts to manufacture cyanides from cyanamide and of the more successful developments of the process in America dating from 1916. In America the growth of the industry has been very rapid, the output for seven months

of the year 1919 being 4,000,000lbs. of a product equivalent to 42% NaCa.

The process as used at present consists of adding a mixture of cyanamide, NaCl and a little CaC_2 (the latter to prevent foaming), continuously to a single phase electric furnace with conducting hearth and suspended electrode. The furnace has small cubical capacity compared to power output, melting is rapid, and the product is removed continuously to a cooling device; quick cooling is necessary to prevent reversion. About 42% more cyanamide than salt is used, and even lower proportions of salt than this have been used satisfactorily. Two brands are marketed of Aero-Brand cyanide, "Grade X" containing an equivalent of 36% to 37% NaCo, and "Grade XX" equivalent to 45% NaCN. In the latter product the cyanogen is said to be present as calcium cyanide.

The application of the material to gold and silver extraction has, it is stated, been perfectly successful. In the case of gold recovery the solutions used are so dilute that the impurities present (chiefly NaCl and free lime) do not cause any trouble. In cyaniding silver ores special dissolving tanks are used, and the calcium salts precipitated and settled out.

Over 90% of the cyanamide nitrogen is recovered as cyanide, and on account of the low price of the raw materials it is anticipated that cyanamide will ultimately be the main source of cyanide and its derivatives.—W. S. LANDIS, *Trans. Am. Electrochem. Soc.*, 1920, Chem. News, December 16, 1921. (H.R.A.)

MISCELLANEOUS.

IRON BACTERIA IN RELATION TO THE INCRUSTATION OF PIPES.—Iron bacteria occur normally in water reservoirs, and predominate in the conduits of these reservoirs. They do no harm in the sense that they give rise to toxic products, but occasionally they multiply in an extraordinary manner and render the water unfit to drink. The problem of greatest interest to engineers lies in the gradual diminution in the bore of pipes. The five species of iron bacteria with which the water engineer should be either directly or indirectly acquainted are *leptothrix ochracea*, *gallionella ferruginea*, *cladotrix dichotoma*, *crenolothrix polyspora*, and *spirophyllum ferruginum*.

A fact which has emerged from a study of the physiology of the iron bacteria is that acid waters are more favourable to the bacteria than alkaline waters.

The iron bacteria can, and do, multiply independently of the presence of iron. They are not iron bacteria in the sense that iron is a necessary part of their food, but if they do multiply in the presence of iron they are capable of furthering the oxidation of ferrous to ferric compounds.

Practical problems centre around four different kinds of growths. These are (1) slimy streamers which appear on the walls and on the bottom of storage reservoirs, and also attached to the internal surface of pipes; (2) tubercular incrustations on the pipes; (3) iron incrustations on non-ferruginous surfaces; and (4) spongy disease of cast iron.

The first are chiefly caused by gallionella and spirophyllum, and although the amount of iron in the water is only one part in a million the increasing accumulation ends in large quantities being collected in the streamers.

With regard to the second, the occurrence of ferruginous tubercles is common and the formation may, and does, take place in the complete absence of the iron bacteria.

The formation of tubercles is dependent on the influence of CO_2 in the water. The influence of iron bacteria is thus accelerative rather than causative, inasmuch as they enrich the immediate neighbourhood of a tubercle with CO_2 .

As to the third problem, the incrustations that form on non-ferruginous surfaces must obviously obtain their iron from the water, and are entirely due to iron bacteria.

Spongy disease of cast iron is not in any way caused by bacteria. Discussing the remedies, it was stated that each water supply must be considered on its own merits.

It must first be ascertained that the organisms concerned are iron bacteria. If this is proved every factor which is favourable to the life of such bacteria is to be removed.

If the organic matter is removed from the water, or oxidised, there will be very little material upon which the bacteria can subsist. Anything tending to reduce acidity will diminish the power of the bacteria to do harm.

There should be periodical biological as well as chemical tests for water reservoirs.

The following tests should be conducted periodically:—

(1) The amount of organic matter, as determined by the number of bacteria per cubic centimetre that the water can support.

(2) The acidity of the water.

(3) The organisms that are established in the reservoirs and conduits, and particularly the iron bacteria that are normally present in the conduits.—DR. DAVID ELLIS, *Water and Water Engineering*, November 21, 1921, p. 420. (R.A.C.)

THE FROTH FLOTATION OF COAL.—Messrs. O. C. Ralston and A. P. Wichmann discuss in this paper research work on the recovery of coal from washery refuse and ashes by means of froth flotation.

The authors point out that although considerable advancement in washing of coal has been made by the employment of ore-concentrating tables, there is still a considerable quantity of what is now deemed refuse hoisted to the surface from which useable coal might be recovered by flotation. In view of the rising cost of coal this is a subject of considerable importance.

The material in a coal seam may contain all gradations from pure coal with almost no ash to pure ash. It has more or less arbitrarily divided into the following types of material: Pure coal, containing up to 5 per cent. ash; bony coal, containing from 5 per cent. to 15 per cent. ash; bone, containing from 15 per cent. to 50 per cent. ash; bituminous shale, containing from 50 per cent. to 80 per cent. ash; and pure shale rock or ash, with very little bituminous matter.

In all the various coal listed it has been found that pure coal is most easily floated, and the other products follow in order,

The suitability of various flotation oils is discussed and the fact noted that although in England the waste water from washing of crude naphthalene was found to be best suited to the work done there, in the United States the most favourable oil was found to be a mixture of alphanaphthalamine (X-cake) with xylidine in sufficient quantity to keep it in solution.

The addition of creosotes, tars or essential wood oils results in the flotation of other portions of the coal.

To produce a clean coal a relatively poor flotation condition is required. Otherwise difficulty is experienced in preventing flotation of other bituminous matter.

Experiments conducted by the Skinningrove Iron Co., Wales, showed that by crushing to — 10 mesh and treating in "minerals separation" machines with naphthalene wash water, ingoing material carrying from 13 per cent. to 45 per cent. ash was reduced to a froth concentrate containing 3.5 per cent. ash. The froth concentrate was thickened in a "Dorr" tank to 50 per cent. solids, and then passed through an "Oliver" continuous filter. This yielded a product containing 10 per cent. moisture which was coked and used for the production of pig-iron at a cost of about three shillings per ton below normal.

A screen analysis of the froth concentrate shows that the — 10 + 20 mesh product contained only 2.7 per cent. ash, while the — 2.0 mesh product contained 5.9 per cent. ash; the tendency was for progressively smaller sizes to be progressively dirtier.

The advantages of the method are set out as follows:—

(1) Ash reduction. Saving in cost of handling, particularly where the place of production of material to be coked is some distance from the coke ovens.

(2) Economy in coking. The product is more highly bituminous, yields richer gas, more by-products and about the same yield of coke.

(3) Superior coke. The coke produced is low in ash and also highly porous, but very strong and dense.

(4) Economy in furnace space. More room for iron ore and fuels.

(5) Minimum silica, since ash reduction calls for less lime and hence less coke.

(6) Dense coke means less fires. This avoids loss of coke fires which must be screened out.

(7) Thermal efficiency. Less slag produced to carry off heat.

(8) Maximum thermal efficiency at smelting zone, because coke does not shatter and burn up before it reaches this zone.

Experiments on the recovery of coal from ash are also referred to, but little data is given.

Tabulated results of experiments carried out on Pacific North-West coals are given, with remarks on the types of machines and oils used, and the disposal of the concentrate.

The conclusions arrived at by the authors are:—

(1) Due to unexplained difficulties, very finely ground coal slime does not concentrate well by flotation, and the process is limited to the treatment of the more granular material. Whereas the finer sizes should give a cleaner concentrate, they actually give a dirtier,

(2) Coals with a bright lustre have been found to concentrate more cleanly than dull coals.

(3) There are indications that cleaner flotation is obtained when the coal is broken into sharp, angular fragments than when ground to rounded pieces.

(4) In general, coals of higher rank, like the semi-anthracite and coking coals, are more amenable to flotation; this is not essentially so, as some of the sub-bituminous coals have been successfully floated.

(5) The mode of occurrence of ash in the coal is highly important. It has been found that most of the ash in bone and bony coal cannot be liberated by the finest attainable commercial grinding.

(6) Clean fine ash can be left in the flotation tailing, and the remainder of the coal easily floated from it.

(7) The cleanest coal is the most easily floated, provided the flotation oil is a poor frother with no collector, and a sub-aeration or pneumatic flotation machine is used.

(8) Bone coal floats almost as easily as clean coal, but requires the use of better frothing oil and some collector for complete extraction.

(9) Bituminous shale is less easily floated than bone coal.

(10) For grading coal by flotation into clean coal, bone, bituminous shale and free ash, careful control and the use of differential methods is necessary.

(11) Flotation tests on removing the ash from finely ground coke have not shown much success.—O. C. KALSTON and A. P. WICHMANN, *Chemical and Metallurgical Engineering*, March 15, 1922. Vol. 26, No. 11, p. 500. (J.E.T.)

THE PHOSPHATE INDUSTRY: SUPERPHOSPHATES.—This is a general survey rather than a scientific treatise. Much of the matter is well known, but some points are of interest. He states, for example, that the manufacturer does not altogether object to considerable quantities of carbonate of lime such as is found in N. African deposits (12-20%) as the rapid evolution of gases promotes more intimate admixture and more rapid combination of the acid and the phosphate and the superphosphate cake is drier and more friable. If superphosphate does not dry while hot it will not dry naturally.

4% of iron and alumina is detrimental, causing reversion; the content of water soluble is being reduced 2-4% within a week of manufacture.

In certain countries it is held that reverted calcium phosphate ($\text{Ca}_2(\text{HPO}_4)_2$) is just as valuable as a plant food as the water soluble (the British standard), but the author seems to use the terms "citrate" and "citric" soluble somewhat indiscriminately.

The article concludes with some remarks of a general nature on superphosphate as a fertilizer, and the author holds the opinion that superphosphate is likely to hold its own even if there were no other reason than that it gives quick return for outlay.—W. PACKARD, *Journ. Soc. Chem. Ind.*, August 31, 1921. (A.W.)

PIPES FOR PRESSURE CONDUITS.—Iron, in the form of cast iron or steel, hitherto almost the sole material used for the construction of con-

duits for conveying water under pressure, has of recent years proved unsatisfactory, particularly for the conveyance of the, generally, ferruginous waters from upland gathering grounds. This is shown in the comparatively rapid diminution in the carrying capacity of such conduits through internal nodular corrosion and slimy deposits.

Nodules originate apparently from the action of CO_2 in the water on the metal, through evidently unavoidable imperfections in all forms of bituminous coating. Possibly also bacterial action contributes to their formation.

Lime treatment and de-aeration have been tried to prevent incrustation, sand filtration to prevent the entry of iron bacteria. These attempts have proved costly and by no means successful. Deterioration of discharging capacity of cast iron and steel pipes has recently been measured to the extent of 25% in 15 years in 42" mains at Birmingham, of 25% in 8 years in Liverpool's 42½" mains, and of 40% in 20 years in Manchester's 44" mains.

The only practical palliative is periodical scraping or brushing out of the mains.

Long observation has shown that good cement concrete surfaces in contact with normal upland waters do not suffer deterioration; that there is no destruction, no nodular or other incrustation, and but little slimy or other deposit.

Certain types of reinforced concrete pipes and of steel tube embedded in concrete have been in use for some years and have proved satisfactory. Water mains of the latter type are in use at pressure up to 500 feet head; the diameters of the pipes being up to 60 inches, working at 255 feet pressure. Of late years there has come to the front a new form of reinforced concrete pipe, in which steel wire reinforcement is embedded in a centrifugally made concrete cylinder. This form of pipe may be used in large diameters up to working pressures of 230 feet. For higher pressures, steel cylinders lined internally—preferably by centrifugal process—with lightly reinforced concrete may be used. Alternatively to steel, the cylinder may perhaps be made of centrifugally cast iron which possesses far greater tensile strength than ordinary cast iron, and is apparently free from the frailties to which the latter is so liable.—F. W. MACAULAY, M.Inst.C.E., *Water and Water Engineering*, November 21, 1921, p. 416. (R.A.C.)

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No. 2.

STATISTICS OF EXTREMELY DEPRECIATED CURRENCIES.

By R. A. LEHFELDT.

Cannan has shown in a recent article* that when a currency becomes extremely depreciated it is no longer true that the elasticity of demand is equal to unity: the demand becomes more inelastic. Statistical material exists for testing this conclusion, and the results are of some interest.

Let Q be the amount of money in circulation P be the value of the unit in an invariable standard (provisionally, say, in gold).

$T = PQ$ be the total value of the currency. Then the elasticity of demand, e , is, by definition

$$e = - \frac{P}{Q} \frac{dQ}{dP} = - \frac{d \log Q}{d \log P}$$

$$\text{therefore } \log \frac{Q}{Q_0} = - \int_{P_0}^P e \, d \log P$$

$$\text{Then } V = \int_{P_0}^P \left(\frac{Q}{Q_0} \right)^{-\frac{1}{e}} dQ = \frac{P_0 Q_0^{\frac{1}{e}}}{1 - \frac{1}{e}} \left[Q^{1 - \frac{1}{e}} \right] = \frac{P_0 Q_0}{1 - \frac{1}{e}} \left\{ \left(\frac{P}{P_0} \right)^{1 - \frac{1}{e}} - 1 \right\}$$

If $e = 1$, V becomes infinite when P is diminished indefinitely which, of course, is equivalent to a proof that e cannot keep up the value unity. If e is notably less than unity the situation is very different; for in the limit

$$\frac{V}{T_0} = \frac{e}{1 - e}$$

so that if $e = \frac{3}{4}$, $V = 3 T_0$ and the most that Government can acquire by inflation is three times the normal value of the currency (i.e. its value when not depreciated).

The values of P and of Q are published in the Monthly Bulletin of Statistics of the League of Nations; the values for Germany, Poland, and Austria have been tabulated and T worked out, and the results are shown in the accompanying graph.

If e is not constantly unity, let us assume as a first approximation that it has some other constant value. Then

$$\log \frac{Q}{Q_0} = - e \log \frac{P}{P_0}$$

$$\text{or } \frac{Q}{Q_0} = \left(\frac{P}{P_0} \right)^{-e}$$

$$\text{whence } \frac{T}{T_0} = \left(\frac{P}{P_0} \right)^{1 - e} = \left(\frac{Q}{Q_0} \right)^{\frac{e - 1}{e}}$$

Now when a Government issues fresh currency dQ to its subjects it extracts from their pockets the value $P \, dQ$ so that the total value obtainable by this method is

$$\int_{Q_0}^Q P \, dQ = V \text{ say.}$$

The first point to be brought out is that the depreciation has been markedly periodic; i.e. it is affected by psychological influences which show a period—of about ten months. Instead of the values of P depending only on the values of Q at the moment, they are affected by anticipation of what Q is likely to become: affected, of course, through the mechanism of speculative purchases and sales. Optimism prevailed about the middle of 1920, in the spring of 1921, and the beginning of 1922, with well marked pessimistic minima between: the period being, in the case of all three countries, very closely ten months. The semiamplitude of the oscillations is round about 25 per cent. of the mean.

The data are not worth analysing by any very elaborate mathematical method, but the

GERMANY.

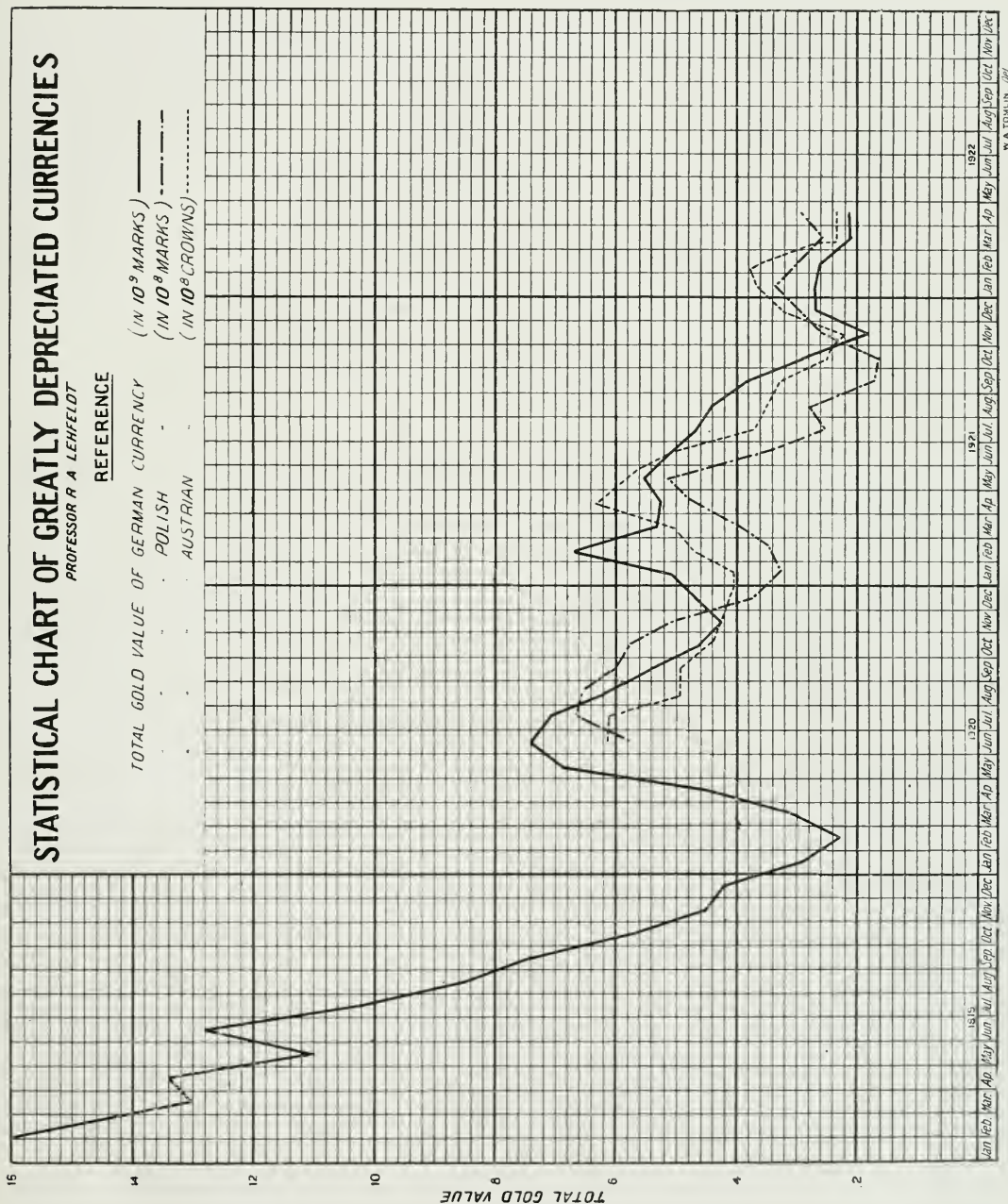
Date.	Berlin Exchange %	Notes in Circulation (10 ⁹ marks).	Gold Value (10 ⁹ marks).
1919 I.	49.2	33.65	16.56
II.	43.4	34.52	14.98
III.	35.8	36.33	13.00
IV.	35.5	37.73	13.39
V.	28.2	38.91	10.97
VI.	30.5	41.91	12.78
VII.	24.8	40.99	10.17
VIII.	21.3	39.98	8.52
IX.	17.8	41.51	7.35
X.	13.5	42.24	5.70
XI.	10.2	43.89	4.48
XII.	8.7	48.57	4.23
1920 I.	5.88	49.5	2.911
II.	4.33	52.8	2.286
III.	5.28	57.9	3.057
IV.	7.42	65.6	4.497
V.	11.22	61.6	6.911
VI.	11.15	66.6	7.426
VII.	10.56	67.9	7.170
VIII.	8.84	70.4	6.223
IX.	7.33	73.8	5.410
X.	6.15	75.3	4.631
XI.	5.53	77.0	4.258
XII.	5.77	81.1	4.679
1921 I.	6.51	78.3	5.079
II.	8.56	78.5	6.720
III.	6.72	79.9	5.369
IV.	6.60	80.7	5.326
V.	6.82	81.2	5.538
VI.	6.07	84.3	5.117
VII.	5.46	86.0	4.626
VIII.	5.01	88.1	4.414
IX.	4.09	94.2	3.853
X.	2.88	99.1	2.854
XI.	1.67	108.5	1.812
XII.	2.15	122.2	2.627
1922 I.	2.19	123.6	2.707
II.	2.04	128.2	2.615
III.	1.52	139.6	2.122
IV.	1.456	149.8	2.181
V.
VI.

POLAND.

Date.	Warsaw Exchange %	Notes in Circulation (10 ⁹ marks).	Gold Value (10 ⁹ marks).
1920 VI.	2.643	21.73	574
VII.	2.516	26.31	662
VIII.	2.119	31.09	656
IX.	1.815	33.20	601
X.	1.505	38.46	579
XI.	1.172	43.24	507
XII.	0.758	49.36	374
1921 I.	0.605	55.08	333
II.	0.549	62.56	344
III.	0.533	74.09	395
IV.	0.546	86.76	474
V.	0.537	94.58	508
VI.	0.344	102.7	353
VII.	0.219	115.2	252
VIII.	0.209	133.7	279
IX.	0.112	152.8	171
X.	0.089	182.8	163
XI.	0.123	207.0	255
XII.	0.130	229.5	298
1922 I.	0.139	239.6	333
II.	0.121	247.2	299
III.	0.100	250.7	251
IV.	0.110	260.6	289

AUSTRIA.

Date.	Vienna Exchange %	Notes in Circulation (10 ⁹ crowns).	Gold Value (10 ⁹ crowns).
1920 VI.	3.432	16.97	616
VII.	3.272	18.72	612
VIII.	2.448	20.05	491
IX.	2.211	22.27	492
X.	1.752	25.12	440
XI.	1.540	28.07	432
XII.	1.343	30.65	412
1921 I.	1.170	34.53	404
II.	1.234	38.35	473
III.	1.224	41.07	503
IV.	1.402	45.04	631
V.	1.276	45.58	582
VI.	1.036	49.68	515
VII.	0.695	54.11	376
VIII.	0.588	48.53	344
IX.	0.468	70.17	328
X.	0.275	90.90	250
XI.	0.182	120.6	219
XII.	0.188	174.1	327
1922 I.	0.1608	227.0	365
II.	0.1431	259.9	372
III.	0.0753	304.1	229
IV.	0.0659	346.7	229



following approximation will serve to determine the magnitude of e .

Taking the case of Austria, the cycle extending 1920 VI. to 1921 IV. has its mean date as 1920 XI. and mean $T = 488$ (unit 10^6 gold crowns): it is succeeded by a cycle whose mean date is 1921 IX. and mean $T = 381$. The values of Q at the two mean dates are 280.7 and 541.1 respectively, so we should have

$$\frac{381}{488} = \left(\frac{541.1}{280.7} \right)^{\frac{e-1}{e}}$$

which yields $e = 0.73$.

Treated in the same way, and over the same period the Polish figures yield $e = 0.67$. In both these cases the value of P was of the order of 1 per cent. or less of the original (or par) value of the money.

German currency has not been depreciated to nearly the same extent, the range in the period considered being between 10 per cent. and 1 per cent. of par. Nevertheless it yields an even lower figure for e —in the neighbourhood of 0.5. The mark has certainly been depressed in value by other than purely commercial influences.

NOTES ON ORGANISATION FOR PROMOTING INDUSTRIAL EFFICIENCY.

By S. DE SMIDT.

One of the terms of reference of the Mining Industry Board is: "The methods to be adopted by the industry to effect economies, to promote efficiency, and to secure the maximum field for employment for Europeans." As this subject is very prominently before us and is occupying the attention of many minds, I make no apology for bringing forward these notes.

To effect economies and promote efficiency is to reduce working costs. This will result in an increase of the scope of mining and a concomitant widening of the field for the employment of Europeans. The great question, therefore, is, how to effect economies and promote efficiency in order to reduce working costs? This question is a very big one, too big to be covered in one paper, because there are so many aspects from which the subject can be approached. The question is best attacked by considering the various aspects separately.

I want to confine myself to a scheme which I came across during my recent visit to the United States, and which, if introduced here and modified to suit local conditions, will, I think, be a great factor in attaining the aforementioned aims.

Whilst in Pittsburg, Penn., I was shown over the iron works of Messrs. Jones and Laughton, one of the largest and most progressive firms in that city. I was struck with the efficient and businesslike manner in which everything was done, and when I remarked upon it I was told of a scheme

in vogue in the works which contributed materially to this attainment.

The scheme is as follows:—

Qualified engineers with the necessary technical training are appointed, one or more for each department. They are called "industrial engineers." They are in attendance at the works, give a hand here and there where necessary, and keep their ears and eyes wide open and their brains active. They study every operation in every minute detail. They study costs, closely watch over and scrutinise each particular item, and compare these with costs of the previous months, and where possible with costs of other factories. Investigations are made where costs are high or on the increase, in order to discover the cause. Faults and leakages are looked for and where possible rectified. They keep in close touch with the very latest practice and developments in the particular branch in which they are interested, and endeavour to keep themselves, and those with whom they come in contact, posted right "up to date." These engineers have no set task or regular routine duties to perform, and are free to concentrate their whole mind on the particular thing needing attention.

The following incident will, in a small way, illustrate the usefulness of such men in the factory:—I was shown through the wire department, and while watching a set of rollers where square bars were hot-rolled into rods of about $\frac{1}{4}$ in. diameter, prior to

being cold drawn into wire, one of the rods fouled the rollers, giving a curious display of red hot iron rod wriggling all over the place. The whole lot had to be scrapped, rolled up and stacked, to be returned later to the smelting furnace. I was then told that it had just been found that this set of rollers had shown an increase in the production of spoilt lots during the previous month. The advantages of the industrial engineer is here seen. The rollers appeared in good working order; the foreman, being responsible for the whole of the wire drawing department, had his full share of worries; the industrial engineer, however, having no responsibility was free to concentrate wholly on this one thing, till the cause of the trouble was found.

These engineers were at first called "efficiency engineers," and had no real status, but it was found that the term "efficiency engineers" raised such prejudice on the part of foreman and men that a change had to be made. They were then called "industrial engineers" and were attached to the assistant general manager's office, and practically ranked as assistants to the Assistant General Manager. This gave them the status needed. They have no authority and cannot give any instruction to foremen or men. The method adopted is that the "industrial engineer" has to work through the foreman. When the "industrial engineer" finds that an improvement can be made, he has to discuss the matter with the foreman, and try to persuade him to see the advantages, and introduce the improvement. This requires some very tactful handling sometimes, for if he antagonises the foreman he finds his efforts baulked, and then it is difficult to make any progress. If he make himself objectionable, the foreman can object to have the particular engineer in his department. Here the engineer learns the art of handling men and getting the best out of them in a peculiarly effective way. If the engineer be unsuccessful in introducing his improvement through the foreman he can then bring it before the Improvement Committee, consisting of the industrial engineers and the departmental engineers, and presided over by the Assistant General Manager. The suggested improvement is then examined and scrutinised, and, if considered advisable, is then further discussed with the foreman by the departmental engineer.

I will deal with this Improvement Committee more fully later on.

These industrial engineers are paid from 125 to 250 dollars a month, and more according to ability. Promotion and increase of salary depend entirely on results. There is consequently a very keen competition among the engineers. I was told that an engineer who did not come forward in any one month with sufficient suggestions for improvements and economies to more than pay for his salary, was in disgrace. There is, therefore, every incentive not to slack, the engineer retaining his job solely by being profitable to the company. The foremen, also, have recognised the value of having the help and suggestions of the "industrial engineers." The system has worked so well, and these engineers have proved their worth to such an extent, that foremen, who in times past have risen from the rank and file, are now picked from these "industrial engineers." The company has recognised the value of getting men as foremen who are trained not only on the so-called "practical side," but men with high technical education. Men, moreover, who have been trained to think and to keep their eyes open, who know where to look for defects and causes of inefficiency, and who have learnt to handle men and make the best use of their inventive and initiative powers. They have realised that a man may be an exceptionally fine welder, but if he has not learnt to think in other directions he does not make much of a foreman. To apply the argument to local conditions, a man may be an exceptionally fine rock breaker, but might make a very poor mine captain. I may mention that this company employs over 10,000 men, and while it can compete in prices with other companies the men are well enough paid to be contented; there is not a single trade union in the factory.

A few words about the Improvement Committee aforementioned and its activities. This committee holds weekly meetings to discuss matters of interest relative to the work in hand, and to receive and examine suggestions for economies and improvements. They give prizes and bonuses, and determine their value according to the value of the suggestions. In this way the men are encouraged to come forward with suggestions, and to make use of what initiative and inventive abilities they have. This has proved to be very profitable to the company. The "industrial engineers"

give assistance to the men in making drawings and putting their ideas before the committee, and take a lively interest in the men's suggestions.

Now for the application of this scheme to these fields. Under present conditions the Manager, Underground Manager, and Mine Captain of a Witwatersrand Gold Mine have, as a rule, their hands so full that they may not be able to give that close and whole-hearted attention to the smaller matters—but it is pennies that make the pound. In the rush and hustle of a gold mine very often bigger matters are overlooked and do not get the attention they deserve, matters which, if rectified, would mean increased efficiency and economy. The scheme could be applied if suitable technically trained men were appointed on a mine, made directly responsible to the General Manager, and these "mining engineers," as they might be called, together with the sectional managers or underground managers, engineer and reduction officer, could form an Improvement Committee, with the General Manager as chairman. Large mines, such as the Crown Mines, E.R.P.M., Randfontein, and Government Areas, could be treated as separate units for this scheme. Smaller mines belonging to the same Mining House may be grouped together. On the large mines mentioned above three or four such men would probably be found suitable; one for the shops, general surface and reduction works, and two or more for underground. If suitable men are obtained one such might give his undivided and concentrated attention to, say, rock breaking by machines; another to the handling and transport of ore; on another occasion attention might be given to the question of handling and transport of jumpers and material—a department in which one sometimes sees a great deal of bungling—and on another occasion the question of ventilation might be tackled, and so on.

The management could offer, through the Improvement Committee, suitable prizes and inducements to all classes of workmen to make suggestions for improvements. This would stimulate a spirit of co-operation between workmen and the management. Let a man realise that his suggestions are worth £ s. d., and he will use his thinking powers, and come forward with suggestions. This is a matter which has not been encouraged on these fields as it might have

been. A central body, like the Mines Trials Committee, did not meet this need. There is need for a local committee in closer touch with local interests. Then again, many men want a little encouragement and help to bring their ideas forward, and such help can be given best by such "mining engineers" as suggested.

The scheme of giving bonuses and prizes for suggestions is in vogue with great success on our railways, as it is, indeed, in a great many large industries all over the world; and why not on our mines? On the Australian Railways the value of the prize is a percentage of the saving effected during the first year. A similar system of payment is under consideration for the Union Railways. The committee of the Union Railways deals with 150 to 200 suggestions a month, some of which are of considerable value, amounting to a saving of hundreds of pounds a year. If the right class of man be obtained for the appointments in the scheme above outlined what a great help they could be to the management, and what improvements could and might be effected in many mines? These men would be a great help to the work of the proposed Improvement Committee.

It is sometimes stated that there seems to have been less progress in mining methods in the mining industry during the last 25 years than in any other industry.

Our mining schools are empty. Does this not mean that the young men, who can afford to spend four or five years on higher education, are choosing other professions where prospects appear brighter? At present the inducement to our best young men to take up mining as a profession is very small on these fields. This will mean a great scarcity, in days to come, of technically trained men on the mines, to the detriment of the mining industry. The "practical man" enthusiast will not agree with me, but what practical man will not be worth a great deal more, if he has a technical education?

If such a career as outlined above is opened it will, I think, encourage young men to come forward to take up mining as a profession, and our mining schools will not long be empty. By means of the above scheme graduates from our mining and technical schools could be trained in the practical working of a mine, would be called upon to think, would have an opportunity

to cultivate their initiative and inventive powers, learn to handle men, and so equip themselves for higher positions.

This scheme can also be applied to our

coal mines, and what a vast field there is in the bye-product industry in connection with the coal mining industry for keen and brainy engineers with new ideas.

INVESTIGATIONS ON THE COPPER-NICKEL ORES OF THE RUSTENBURG DISTRICT.

By J. A. ORTLEPP.

The copper-nickel ore from Vlakfontein, Rustenburg, has been the subject of discussion in a bulletin entitled "Nickel," by Mr. T. G. Trevor, Inspector of Mines, Pretoria.*

Samples.

The samples representing ore from Vlakfontein amounted to about 40 pounds weight, of which about 5 pounds represented oxidised ore.

A large lump of fresh ore was kept in reserve, and the remainder as well as the oxidised ore broken down to two inch cubes by means of hand hammer. Representative portions of each type were retained for microscopic examination, and the remainder crushed to pass a $\frac{1}{4}$ in. mesh sieve. The crushed portions of oxidised and fresh ore were well mixed and treated as one in later investigations. This sample was called ore original in all subsequent investigations, and was crushed to pass a 90 linear mesh sieve.

This mixed aggregate of ore was used till the supply ran out. After that the lump ore kept in reserve as mentioned previously was utilised. This portion was also sampled, and on analysis showed a somewhat different composition, which accounts for the different analyses of ore charges in tests conducted on this material.

Microscopic Examination of the Ore.

Preparation of Specimens.—The specimens examined under the microscope represented oxidised ore (one slide) and fresh ore (six slides). The original samples in each case were about 2 in. in size, with the exception of one piece of fresh ore about 4 in. diameter.

In this investigation the author followed the methods described in the book "Microscopic Examination of the Ore Minerals," by W. Myron Davy and C. Mason Farnham, as far as possible. The investigation aimed chiefly at the determination of the opaque constituents of the ore, and no

transparent slides were prepared to determine the gangue minerals. However, a fairly accurate conclusion with regard to the gangue minerals could be drawn from examination of the opaque slides.

The preparation of the specimens for examination consisted of the following operations:—

1. The rough piece of ore about 2 in. in size was carefully chipped down to about 1 in. size. It was then ground down on one side on a rapidly revolving emery wheel to an area of about $\frac{3}{4}$ in. square. The size of the specimen was now reduced till it could be placed in the barrel of the mounting tube. The thickness was reduced to about $\frac{3}{8}$ in., which size was found to be most convenient to handle while grinding and polishing. As the ore under examination was rather brittle all sharp edges were bevelled off in order to prevent breaking during subsequent grinding and polishing.

2. The flattened face of the specimen was further ground down on a horizontally revolving iron disc fed with a paste of coarse emery and water. The disc was driven by means of a treadle wheel at about 300 r.p.m. Excessive pressure on the ore was avoided as it tended to break up the surfaces of the minerals. When all marks on the flattened surface caused by the emery wheel had been removed the specimen was further ground under very light pressure. The wheel and specimen were now well washed with water in order to remove any particles of adhering emery, and grinding further continued using a finer grade of emery as abrasive. Very light pressure was used, and a few minutes' grinding proved to be sufficient. The specimen was washed to remove all particles of the abrasive sticking to it.

3. Further grinding was conducted on a plate glass slab which was well cleaned and coated with a thin paste of fine emery and water. The specimen was placed face down on the plate and gently rubbed with a rotary motion. Grinding continued thus

till the roughness produced by the last powder used had been removed. At the finishing stages the pressure on the mineral was made as light as possible. The specimen was again washed to free it from the grinding powder. The last stage in grinding was now reached, and was conducted in the same manner as the above, but on a separate glass plate and using a paste of emery dust and water as abrasive. Pressure was kept as light as possible.

4. The final and most difficult step in the preparation of the specimen was now arrived at, namely, polishing. Several methods were tried, but the following gave the best results in this case. A wooden wheel with a plane surface was mounted on the emery wheel spindle, and on its surface was stretched a piece of dry wash leather. The wheel was made to revolve at a high speed and fed with emery dust while the specimen was gently held against it. The emery dust was best applied by means of a small brush.

The surface which was previously dull gradually brightened while polishing proceeded. Pitting was liable to take place if polishing continued too long, so the specimen was examined under a low power at intervals while polishing proceeded. The process was stopped when the specimen had the best polish obtainable without undue pitting. The specimen was now dusted and given a final polish by hand on a block on which wash leather, treated with dry polishing rouge, was stretched.

All the specimens were treated in this manner, and although the polish produced was by no means perfect the relations between the various constituents of the ore were sufficiently brought out to allow of their identification. The specimens were mounted on glass slides in the usual manner adopted for metallic sections. The large specimen mentioned in the beginning of the section was polished in the same manner as the smaller specimens. The final polish obtained, however, was not so good on account of the larger area which had to be treated.

As seen in a hand specimen the ore consists of a massive aggregate of sulphides in which has been included numerous small dark crystals of some other mineral. The ore is heavy and breaks quite easily under the hammer. The photograph (No. 1) shows an unpolished surface of the ore full size. The two main constituents are easily discernible in the photograph. Photograph

No. 2 shows a polished hand specimen full size. In it the relation between some of the constituents becomes evident. The dark mineral occurs in well-shaped crystals, and everywhere has a glassy appearance. It is much harder than the minerals forming the remainder of the section, and in all cases where it occurs as individual crystals it is completely surrounded by sulphides. Cross-sections of this mineral, as seen in the photograph, e.g., inside the white square, as well as in the other specimens, are not unlike those one finds in orthorhombic pyroxenes. This fact as well as the occurrence of the deposit in the norite pyroxenite belt of the Bushveld complex leads one to believe that this glassy mineral belongs to the pyroxene group, and in composition is probably a ferro-magnesian silicate. This mineral, however, will simply be referred to as the gangue in the account that follows.

On examining the sulphide portion of the ore in photo No. 2 lighter coloured patches can be seen in it in several places. These patches are evidently of different composition, and their identity cannot be established until closer examination has been made. Another mineral has also been brought into view (within circle photo No. 2); it occurs in lesser quantity, and is nearly completely enclosed in gangue mineral.

Examination of the specimens prepared for the microscope was now proceeded with. The microscope used was of the metallographic type with vertical illuminator. Each specimen was examined without the application of any reagent to the surface to see if any further identifications could be made. The low power was used for this work. All the specimens appeared to contain the same group of minerals, although in some specimens one constituent would be more plentiful than in others. The specimen representing oxidised ore, however, appeared to be made up of one constituent only, namely, the darker coloured sulphide with several voids filled with a granular material. The gangue and the lighter coloured sulphides were absent. The absence of the gangue in this specimen was due to the fact that it was not present originally when the ore was oxidised, and not to a process of solution during the weathering of the ore. If the latter had been the case then cavities similar in cross section to this mineral as seen in photo No. 2 would have been seen. The gangue is the most stable mineral in the ore under atmospheric conditions, and

if it had been present in this section it would have been the last to be decomposed. The lighter coloured sulphides probably occupied the voids before weathering took place and had been removed by the action of surface waters.

All the minerals to be found in the specimens could be distinguished by their colour as well as the polish they took. The lighter patches in the sulphide portion were examined and found to be made up of a pale bronze coloured mineral. The main body of sulphide was of a dark bronze colour, and did not polish so readily as the lighter sulphide. A few small patches of a bright brass yellow mineral were also to be seen. Two other minerals in much smaller amount were also seen. One of these was of a dull grey colour and did not polish easily; it occurred in isolated but rather large patches. The other mineral occurred in two exceedingly small patches, roughly square and measuring about one-hundredth of an inch in length. This mineral polished well, and had a silvery white colour.

The minerals observed in the sections were as follows:—

- (1) Darker bronze yellow mineral.
- (2) Lighter coloured bronze yellow mineral.
- (3) Bright brass yellow mineral.
- (4) Gangue.
- (5) Dull grey mineral.
- (6) Bright silvery white mineral.

Micro-chemical tests were now applied to the bigger surfaces exposed in order to fully determine the minerals. The reagents used were as follows:—

1. Cold nitric acid (diluted 1:1 in water).
2. Cold hydrochloric acid in water 1:1.
3. Potassium cyanide solution 10 per cent. in water.
4. Di-methyl-glyoxime in alcohol.

The solutions were applied by means of a capillary pipette and the reactions observed under the microscope. The reagents were applied in the order stated, and on all the minerals observed previously. The reactions observed as well as the physical properties of the various constituents are tabulated as follows:—

Mineral No. on Photos	1	2	3	5	6
Colour	Dark Bronze Yellow	Yellow Light Bronze	Bright Brass Yellow	Dull Grey	Bright Silvery White
Polish	Brittle	Well	Well	Brittle	Very well
Hardness	Medium	Medium	Medium	High	—
Other Properties	Magnetic	—	—	—	—
Action with HNO ₃ Aq.	Neg.	Tarnish with development of cracks	Neg.	Neg.	Rapidly went black with effervescence
HCL Aq.	Neg.	Neg.	Neg.	Neg.	Reacts slowly with no effervescence
KCN Aq.	Neg.	Neg.	Neg.	Neg.	Neg.
Di-methyl Glyoxime	Neg.	Red ppt.	Neg.	Neg.	Red ppt
Special Tests	Neg.	—	—	Boiling Aqua regia-Insol.	—

The ore examined was thus made up of the following minerals, which were found to conform most closely with the properties and reactions observed:—

Mineral No.

on Photo.	Mineral.	Composition.
1	Pyrrhotite	Fe ₇ S ₈
2	Pentlandite	NiFeS
3	Chalcopyrite	CuFeS ₂
5	(Unidentified)	—
6	Maucherite	Ni ₃ As ₂
4	Gangue	(pyroxene group)

Microphotographs were taken of portions of the samples which best illustrated the relations between the various constituents, and the following photographs were obtained.

Photo No. 3 is a section which has been acted on by dilute (1:1) cold nitric acid. Pentlandite has been stained while chalcopyrite and pyrrhotite have not been acted on. The minerals are numbered as follows:—

(See list given above.)

The magnification in the photograph is 25 diameters.

Photo No 4 shows a section including the mineral Maucherite (bright patch between two crystals of gangue indicated by No. 6). This section serves to show how well this mineral polishes in comparison to the other sulphides (25 diameters). Photo No. 5 shows an enlarged view of the mineral Maucherite (200 diameters). It shows a peculiar intergrowth at its junction with the other minerals.

Photo No. 6 represents the unidentified grey mineral (No. 5), and its relation to the gangue (25 diameters magnification).

A brief summary of the observations made on the ore sections is as follows:—

Oxidised ore:

1. Only mineral observed is pyrrhotite.
2. Voids containing granular material.

Fresh ore:

1. Chief constituents are: Pyrrhotite, Pentlandite, Chalcopyrite, and a Pyroxene (gangue).
2. The Pyroxene occurs in well defined crystals completely surrounded by sulphides.
3. Pentlandite in general occurs in well defined crystals sometimes completely surrounded by pyrrhotite.
4. Chalcopyrite was always seen in contact with gangue minerals.
5. Pyrrhotite seems to have filled in the spaces between the other minerals.
6. No secondary minerals were seen and evidence of replacement was absent.

Bearing these observations in mind one is led to conclude that the ore of which the sections were samples was formed by a process of magmatic differentiation, and

Mineral.	Sp. Gravity.	Hardness.
Gangue	3.15—3.60	5.50
Pyrrhotite ...	4.58—4.64	3.5 —4.5
Pentlandite ..	4.69	3.50—4
Chalcopyrite .	4.10—4.30	3.5 —4

(Above from Dana's Mineralogy.)

An examination of the specific gravities of the constituents shows that the three sulphides are practically of equal density, while the density of the gangue is unusually high and not far below that of the other constituents. The only separation which can be expected by means of appliances built to utilise the difference in specific gravity of minerals, even if perfect, which is very doubtful under the circumstances, must needs be low on account of the comparatively small bulk of gangue compared

that the order of separation of the minerals from the molten magma was as follows:—Pyroxene (gangue), Pentlandite, Chalcopyrite, Pyrrhotite.

However, it must be borne in mind that the ore examined was not representative of the whole ore body, hence the conclusions arrived at cannot justly be applied to the latter.

The origin of the Vlakkfontein ore can only be fully established by accurate examination of samples from all portions of the ore body. A noteworthy point in connection with the Vlakkfontein deposits is the evidence of igneous activity in their neighbourhood. The Vlakkfontein deposits lie about 10 miles south-west of the great eruptive volcanic mass of the Pilandsberg.

Metallurgical Application of the Microscopic Examination.

The microscopic examination has so far only been used in identifying the mineral constituents and establishing the relation between them. Valuable conclusions which throw much light on the metallurgical treatment of the ore can be drawn from the results thus obtained.

The metallurgical treatment aims at the ultimate isolation of one or more of the valuable constituents of the ore. In this case these constituents are represented by the nickel and the copper present. A knowledge of the manner in which these constituents occur is thus of great value in determining a suitable treatment for their removal.

The following table gives the chief physical properties of the minerals which have been determined by the microscope.

Remarks.
Crystals hard, tough, and elongated.
Magnetic and brittle.
Non-magnetic and brittle.
Non-magnetic and brittle.

with the sulphides. (See photo No. 2.) This cause taken together with the high density of the gangue makes separation by means of gravity extremely difficult and impracticable.

On examining the hardness of the various constituents one finds that the gangue is considerably harder than the sulphides. Crushing the ore to a small size would thus lead to the production of a greater portion of fines from the sulphides, while the hard

constituent will concentrate in the coarse product.

Pyrrhotite forms the greater portion of the valueless material in the ore, and is magnetic. Concentration by magnetic means should thus remove the pyrrhotite and leave the non-magnetic portion with a richer nickel and copper content. However, pyrrhotite sometimes contains a certain amount of nickel, and this together with the adhering particles of nickel and copper minerals would be lost in the magnetic portion.

The physical composition of the ore points clearly to the impracticability of ordinary flotation concentration when applied to this ore. If selective flotation were possible then good concentration of the nickel and copper could be expected.

That the ore is amenable to leaching is borne out by the fact that the sample representing oxidised ore contained no nickel and copper sulphides, but only cavities filled with a granular material. The solubility of these minerals under the action of surface waters is also mentioned in the appendix describing the deposits.

Concentration by means of a smelting process cannot be discussed in the light of physical analysis only. A chemical analysis of the ore must also be known before the trend of a smelting process can be followed. Various concentration tests were conducted on the ore to demonstrate the truth of the conclusions drawn from the physical analysis and the results obtained are set out in the account that follows.

Metallurgical Investigation.

As already stated the greater portion of the ore was represented by a sample described previously. An analysis of this sample was made and gave the following result:—

Sample Ore Original.

SiO ₂ and insol. (in Aqua Regia) ...	24.56%
Sulphur ...	26.06%
Fe ...	39.48%
Ni ...	3.71%
Cu48%
CaO ...	Trace.
MgO ...	1.05%
	<hr/>
	95.34%

The analysis falls short by 4.66%. The following metals were tested for qualitatively but could not be detected: As, Sb, Cd, and Mn. However As was found to be present in the physical examination of the ore in very small quantity in the form of

Ni₃As₂. In an area of about 3 sq. in. of ore examined only 2/10,000 sq. in. of this mineral was to be seen. This seems to point that the percentage of As present was exceedingly small, and, besides making very little difference to the analysis of the ore was present in such small amount as to make its detection by Marsh's method impossible.

Part of the deficiency in the analysis can be attributed to the following causes:—Of the iron present a portion exists as FeO in the gangue mineral. Moisture and Al₂O₃ were not determined, and might possibly have been present. Cobalt was not determined.

Assuming all the sulphur to be combined with all the Ni, Cu, and a portion of the Fe present in the form of the following minerals FeNiS, FeCuS₂, and Fe₂S₃, the remainder of the iron will be present as FeO in the gangue, which was probably a ferromagnesian silicate. Recalculating the analysis on this basis the following result is obtained:—

SiO ₂ and insol. ...	24.56%
S ...	26.06%
Fe ...	32.33%
FeO ...	9.20%
Ni ...	3.71%
Cu48%
CaO ...	Trace.
MgO ...	1.05%
	<hr/>
	97.39%

Concentration Tests.

The ore was now experimented upon by applying various methods of concentration to it in order to see what recoveries were obtainable in each case.

(1) Grading Analysis.—This test was conducted on a portion which appeared to contain a high percentage of gangue constituents. The sample was crushed as follows:—

1. By hand hammer to about $\frac{3}{4}$ in. size.
2. By jaw breaker to about $\frac{1}{4}$ in. size.
3. By a small disc sample grinder set to produce a -10 product.

The crushed product from the third stage contained several elongated crystals of gangue which remained uncrushed and did not pass a 10 mesh sieve. A representative portion of this sample was graded, using the following screens: 10 mesh, 20 mesh, 30 mesh, and 50 mesh. The results obtained are as follows:—

Results and Balance Sheet.

Material.	% Wt.	% SiO ₂ .	% Ni.	% Ni. re-covered.
Ore	100.00	41.06	2.89	—
+ 10	4.82	71.31	.85	1.40
- 10 + 20	34.33	48.89	1.66	19.80
- 20 + 30	15.30	37.64	2.35	12.40
- 30 + 50	18.80	31.34	3.76	24.50
- 50	25.90	27.16	4.51	40.30
	99.17			98.40

The grading analysis fully bears out the first conclusion drawn from the physical analysis of the ore, namely, that the softer minerals, e.g., Ni, will be concentrated in the fines, while the harder gangue will tend to remain in the coarser products. The grading analysis is valuable from a metallurgical point of view where a large quantity of fines are produced in conjunction with the smelting process adopted. A certain quantity of the fines could be removed by screening without seriously affecting the percentage extraction, while at the same time increasing the grade of the material treated and decreasing the bulk.

(2) Wilfley Table Concentration (Gravity).—This test was conducted on a portion of the crushed ore, designated by the sample as ore original, and containing 3.71% of nickel. A small laboratory wilfley table was used, and the slope and speed were so adjusted as to give the best concentration possible. Two products were finally obtained, and the sample which was crushed to pass a 30 linear mesh sieve was ungraded before concentration. The following results were obtained:—

Results.

Material.	% Weight.	% Nickel.	% Recovery Nickel
Ore	100.00	3.71	—
Concentrate	17.80	4.9	23.80
Tailing	82.10	3.43	76.20

The low recovery recorded indicates that gravity concentration is impracticable, a conclusion arrived at in the microscopic examination of the ore.

(3) Magnetic Concentration.—Magnetic concentrations were next conducted on the ore both in the raw and roasted states. The roasted ore was subjected to a slight roast in a wide, open crucible in a wind furnace. The method of conducting the concentration was as follows:—

The sample to be tested was ground to pass a 30 mesh screen and two 100 gram portions weighed out. One portion was roasted as described above, and allowed to

cool. Two clean sheets of brown paper were laid on a table next to each other, and one of the samples spread in a thin layer on one of the sheets.

The magnetic portion was now removed by slowly passing a large horseshoe magnet over the surface of the ore but not in actual contact with it. The magnetic portion of the ore clings to the magnet, and when sufficient has accumulated the magnet is trapped with the hand to dislodge any non-magnetic minerals from the adhering particles. The particles adhering to the magnet are brushed off on to the other sheet of paper. The whole surface of the ore is thus carefully and slowly traversed, and all magnetic material removed to the other portion of the sample. The non-magnetic residue is scraped together, well mixed, spread out, and traversed in the same manner to ensure that all magnetic minerals have been removed. The magnetic portion is now similarly traversed. A non-magnetic residue remains, and is added to the non-magnetic portion. This is repeated till the non-magnetic residue finally disappears. Both roasted and unroasted ore are treated in the same manner.

The products were assayed with the following results:—

Unroasted Ore.

Material.	% Wt.	% Ni.	Recovery % Ni.
Ore	100.00	3.71	—
Magnetic	61.10	3.33	54.80
Non-magnetic ...	36.40	4.94	48.50

Roasted Ore.

Material.	% Wt.	% Ni.	Recovery % Ni.
Ore	100.00	3.71	—
Ore roasted	92.90	—	—
Magnetic	67.30	3.29	59.60
Non-magnetic ...	24.80	5.91	39.50

Although the recoveries in the concentrate were higher in both cases than they were in the case of gravity concentration the process cannot be considered an economic success as only 39 to 48 per cent. of the total nickel is accounted for in the richer products. The physical condition of the ore again pointed to the facts which have been borne out by this test.

(4) Flotation Tests. — Three flotation tests were conducted on samples crushed to -90 mesh. The conditions under which each test was conducted and the results obtained are set out as follows:—

Test No.	I.	II.	III.
Vol. water used, C.C.	2000	2000	2000
Wgt. ore used, gms.	500	750	750
% Oil (wood creosote)	7	7	7
Temp. water degrees, C.	34	34	34
Time agitated mins.	10	10	10
H ₂ SO ₄ in circuit	0	12 C.C.	0
Froth	Poor	Good	Moderate

	Head.	Tail.	Head.	Tail.	Head.	Tail.
Wgt. recovered gms.	40	—	285	—	121.5	—
% Wgt. recovered	8.0	92.0	38.0	62.0	16.0	84.0
% SiO ₂ in products	7.19	27.24	4.93	39.0	7.23	29.08

SiO₂ concentrates in the tailing, but as it only makes up about 25% of the bulk of the ore, concentration of nickel cannot be high even with perfect separation.

Matte Smelting.

Concentration by physical means having proved to be unsuccessful, smelting to a matte was attempted to see if better results could be obtained, and which smelting charges were most satisfactory.

The charges were smelted in crucibles heated either in wind furnaces fired with coke or in a small blast furnace fired with gas retort carbon. In the former case the smelting temperatures ranged between 1,100 and 1,200 degrees centigrade, while in the latter case they rose to about 1,300 to 1,400 degrees.

As it is difficult to obtain quantitative results in work of this description when working on a laboratory scale only one example of these matte smelting experiments is included.

The charge in this case was made up as follows:—Ore used 200 gms., of this 170 gms. were roasted and 30 gms. unroasted. The sulphur content of the roasted portion was not determined. 68 gms. of SiO₂ (dump sand) were added in order to slag with the excess of FeO in the charge. The figures reproduced above were taken for the following reason: The 170 gms. of roasted ore were assumed to be practically free from S, and all the sulphur for the matte was to come from the unroasted 30 gms. of ore. The gangue in the ore was assumed to be an iron magnesian silicate, and the 68 gms. of sand were added in order to slag with the FeO from the 170 gms. of roasted ore to produce FeSiO₃.

The charge was melted down in a fireclay crucible in a wind furnace at a temperature of about 1,100 to 1,200 degrees C. It took two hours to fuse, and by that time the slag which had formed was very pasty. Attempts to raise the temperature of the wind furnace were without result, and accordingly

10 gms. of CaCO₃ were added to the crucible charge with the object of making the slag more fusible. A slight improvement resulted, and after about half an hour's further heating the charge was poured into a conical mould. The slag was still viscous and flowed with difficulty. When cold a small button of matte was recovered and the slag on being broken contained several shots of matte. 41.5 gms. of matte were recovered; the slag was not weighed. The products of the smelt were crushed and analysed and the following results obtained:

Charge: Unroasted ore ...	30 gms.
Roasted ore	170 gms.
Sand	68 gms.
CaCO ₃	10 gms.
Matte recovered	41.5 gms.

Analyses of Products.

	Ore.	Matte.	Slag.
SiO ₂ and In-sol....	24.56	.72	43.50
Sulphur	26.06	27.60	—
Fe	32.33	51.80	—
FeO	9.20	—	45.90
Ni	3.71	14.58	.50
Cu48	1.25	—
CaO	—	—	5.37
MgO	1.05	—	3.2

The distribution of nickel in the products was as follows:—

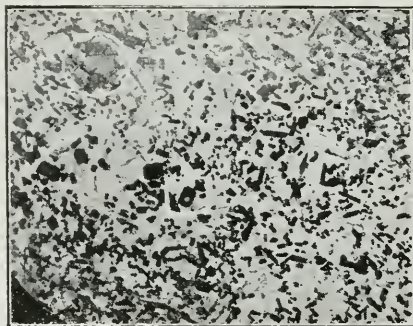
Nickel: Per cent. Recovery.

Matte	81.80
Slag (difference) ...	18.20

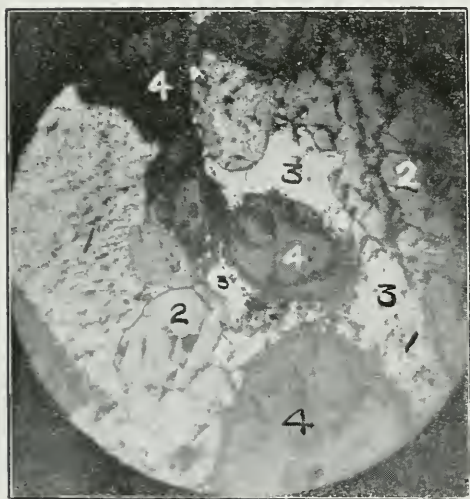
The charge was so calculated as to give a matte running about 30% nickel and copper, and, as only 15.83% of these metals were contained by the matte it can only be concluded that the sulphur content of the charge was too high. The slag produced was viscous and contained shots of matte. Several of the latter were removed and weighed with the matte. The Ni. content of the slag was still on the high side. This was probably due to the inclusion (mechanical) of matte in the viscous slag.



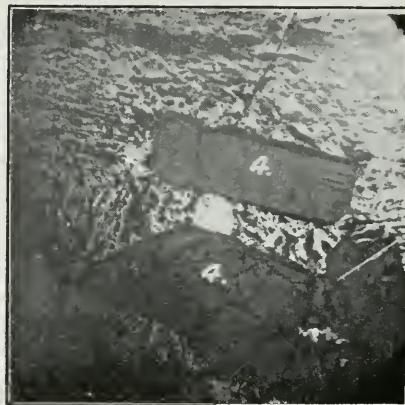
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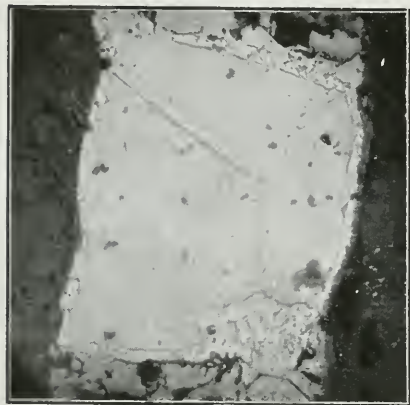
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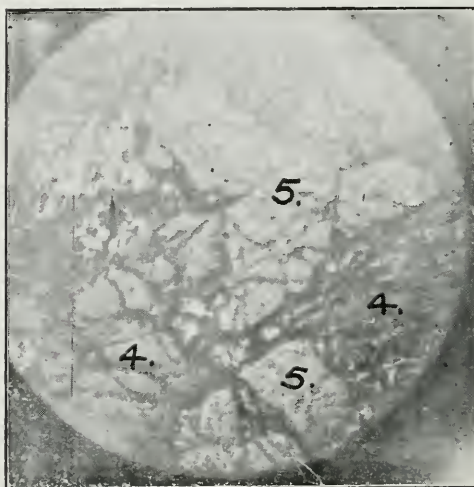
No. 3.



No. 4.



No. 5.



No. 6.

The grade of the matte produced in these tests being somewhat low, viz., 9 to 15% nickel, attempts were made to increase it. The matte was oxidised by blowing air through it while in the molten state, and fluxing any oxides formed by adding SiO_2 from time to time. The principle was the same as that adopted in practice, although the grade of matte was far below that obtained in practice.

The matte was heated in a fireclay crucible standing in a larger crucible. The flame from two gas burners was allowed to burn between the two crucibles, and provide the heat for keeping the matte and resulting slag in the molten state. Air was blown through the molten matte by means of a clay pipe tuyere held in a long piece of bent gas piping.

300 gms. of low grade matte were placed in the inner crucible and heated to the melting point. Air was now slowly blown through the matte and a little sand added at intervals. The SiO_2 slagged readily, but the temperature had to be kept well above the M.P. of the matte to keep the slag in the molten state. In all 137 gms. of sand were added to the crucible, and as the mass of slag increased it became more difficult to keep the slag liquid. A little lime was added to the crucible in order to reduce the M.P. of the slag. While refining was proceeding matte was blown out of the crucible in the form of small globules and sparks. Blowing was continued till the volume of matte had diminished appreciably. The tuyere was removed, the cover placed on the surface, and the whole heated up in order to completely fuse the slag. After some time the charge was poured into a button mould, and 39 gms. of matte were recovered. This great reduction in wt. led to the suspicion that losses besides those due to the oxidising power of the blast were present. An examination of the crucible and the furnace when cold showed that although the former was perfectly intact a considerable portion of the matte as well as slag had passed through the crucible by way of the pores and crevices. An analysis of the products obtained showed an increase in the nickel content of the matte from 14.60% to 25.30%.

The conclusion drawn from the various smelts conducted was that concentration of the nickel into the form of matte is possible. The sulphur is the only reducing agent in the ore, and the low grades of matte produced were due to insufficient roasting. In cases where the sulphur content was lower

in the charge the matte produced was higher in nickel. Less sulphur means less iron reduced, and thus a higher value for nickel in the matte. If smelting were conducted on this ore in blast furnaces then a considerable portion of this unroasted sulphur would be oxidised in the furnace, and a higher grade of matte produced.

Any precious metals present in the ore are concentrated in the matte, and will undoubtedly be concentrated in the refined matte if refining is practised. Metals of the platinum group are present, although their identity was not fully established.

In conclusion the author wishes to take this opportunity of expressing his gratitude to Mr. Chudleigh, who had so kindly provided him with the ore samples. Further he wishes to express his indebtedness to Prof. G. H. Stanley, Mr. H. R. Adam, and Mr. D. Millin for the valuable assistance rendered him during the course of the investigation.

FEATURES OF METALLURGY OF A REFRACTORY GOLD ORE.

By F. WARTENWEILER.

(*Printed in Journal, February, 1922.*)

DISCUSSION.

Mr. J. Hayward Johnson: Mr. Wartenweiler's paper is especially interesting, in that it details the experiments and work carried out on a very refractory ore by one of our earliest Presidents and our immediate Past President.

In South Africa ores of a similar character undoubtedly exist, but unfortunately, except in a few instances, has any serious attempt been made to treat them, as most of these propositions have been in the hands of individuals or small syndicates who have not had the capital to expend on experiments. The practice generally pursued has been to erect a plant suitable for treating a simple ore, which generally exists in the oxidised portion of the mine, but once this zone is passed a sulphide ore is encountered (as at Prestea), trouble in recovery arises, losses occur, and the mines have been abandoned as unpayable.

Mr. Wartenweiler's description of the operations and interesting flow sheet of the Prestea Block A plant demonstrates very clearly that ores of such refractory nature can be successfully treated.

State assistance in conducting experiments and giving advice to workers of propositions

of this nature would no doubt be heartily welcomed and considerably assist in opening up new fields or reopening old fields of gold mining industry in South Africa.

Laboratory work alone does not give all the information required. It is often only under working conditions that the true refractory nature of the ore is established. Freshly prepared solutions will give good results, but they gradually become fouled and the extraction of gold decreases, as experienced at the Machavie G.M. Co. (see page 256, Vol. XX.).

One noticeable feature in treating the Machavie ore was that the mill water became opaque, although all solids were settled out. Precipitation was good, but a large amount of white precipitate was formed.

BOOK REVIEW.

COAL. By Elwood S. Moore, M.A., Ph.D.
London: Chapman & Hall. New York:
John Wiley & Sons. Pp. 462. Price,
25/- net.

This work deals with coal, its properties, analysis, classification, geology, extraction, uses, and distribution. The author is professor of geology and mineralogy, and Dean of the School of Mines of the Pennsylvania State College. It seems almost impossible to co-ordinate all the above information within the covers of a single book. The author has very successfully achieved what he set out to do, and has exercised very wise discretion in his treatment of the subject under the various heads. Naturally he has not been able to exhaust the subject under each department, but the result has been a very useful volume which will serve as a valuable reference for those interested in coal.

The illustrations, both photographs and plates, are particularly clear and instructive.

G.A.W.

Notices and Abstracts of Articles and Papers.

CORRIGENDUM.

JULY, 1922, page 13. The first two lines of the Abstract near foot of page should read: "Determination of Alkali Hydroxide and Carbonate in Presence of Cyanides. A."

CHEMISTRY

APPLICATION OF THE ELECTRON THEORY OF CHEMISTRY TO SOLIDS.—Each kind of atom has associated with it a definite number of electrons which form its outer layer when it is in the free state; it is by the re-arrangements of these electrons that it is able to hold other

atoms, whether of the same or different kinds, in chemical combination. When these atoms aggregate and form a solid there will be in each unit volume of the solid a definite number of these electrons, and the problem is to distribute the electrons so that they will form with the atoms a system in stable equilibrium. . . . We shall begin with the simplest case when the atoms are all of the same kind, i.e., when the solid contains only one chemical element. We suppose that the electrons are arranged as a series of cells which fill space and that each cell surrounds an atom; the number of cells is equal to the number of atoms. If the atom is monovalent the number of electrons is equal to the number of atoms, if divalent to twice that number, if trivalent to thrice that number, and so on. This condition will determine the shape of the cell. If the cells have to be similar and equal and to fill up space without leaving gaps, they must be of a limited number of types. These are as follows: (1) Parallelepipeda, if the atoms are of the same kind these may be expected to be cubes; (2) hexagonal prisms; (3) rhombic dodecahedra; (4) cubo-octahedra.

Let one illustration suffice for the type of reasoning employed. Suppose space to be filled with cells of cubical form packed one against the other, and at each of the eight corners of each cube let there be an electron, and, in addition, let there be an electron at the middle of each of the twelve edges of the cube. First, we consider the corner electrons alone. At each corner eight cubes meet and one electron is there, one-eighth of an electron to the account of each cube. The same holds for each of the eight corners of each cube. To the cube's account we therefore reckon eight times one-eighth or one entire electron. Now for the edge electrons. Only four cubes come together at this edge, so the electron at the middle of this common edge is divided equally between the four cubes. The cube has twelve edges, so to its account from the edges there are twelve times one-fourth or three electrons. One electron from the corners and three from the middles of the edges make a total of four for each cube, a possible arrangement for a tetravalent element, which turns out to be less stable than another arrangement. "For most of the valencies more than one arrangement of the electrons is possible, indicating that for such elements there might be allotropic modifications with different crystalline forms." The author next examines on dynamic grounds the stability of certain of the possible configurations of the electrons. This leads to a calculation of the maximum frequency of the vibrations of the electrons and some very satisfactory agreements between the outcome of theory and the data of experiment are noted. Similar agreements are further found in the cases of the bulk modulus and of the dielectric constant.

This paper marks a notable advance toward the complete understanding of crystals.—SIR J. J. THOMSON (*Phil. Mag.*, April, 1922), *Journal Franklin Institute*, June, 1922, p. 861. (J.A.W.)

CHANGES OF ADDRESS.

EXTER, J. K., 1/o Tsumeb; 159, Rosetta Street, Pretoria West.

IRELAND, H. J., 1/o Johannesburg; P.O. Box 1260, Capetown.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA

+ +

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Vol. XXIII.

SEPTEMBER, 1922.

No. 3.

Proceedings

AT

Ordinary General Meeting, 16th September, 1922.

The Ordinary General Meeting of the Society was held in the Assembly Hall, Scientific and Technical Club, 100, Fox Street, Johannesburg, on Saturday, the 16th September, 1922, at 8 p.m., Prof. G. A. Watermeyer (President) in the chair. There were also present:—

26 Members: Messrs. H. R. Adam, C. J. Gray, F. W. Watson, J. R. Thurlow, J. Hayward Johnson, A. King, Dr. A. J. Orenstein, J. M. Thorburn, F. Wartenweiler, John Watson, A. Whitby, H. A. White, J. L. Willey, J. A. Woodburn (Members of Council), H. D. Bell, Dr. W. A. Caldecott, J. Chilton, B. J. Hastings, J. Henderson, E. Homersham, J. D. Marquard, Prof. H. H. Paine, H. Pirow, T. K. Prentice, R. A. H. F. de Smidt, and H. R. S. Wilkes.

6 Associates: C. E. Deakin, S. Evans, O. A. Gerber, S. K. Gilbertson, F. W. Lethbridge, and W. Simpson, and
2 Visitors.

MINUTES.

The President asked that the Minutes of the Ordinary General Meeting and the 26th Annual General Meeting, held on the 20th May and 24th June last, respectively, as published in the *Journals* for those months, be confirmed. This was agreed to.

NEW MEMBERS.

A ballot was taken for the election of new members, and the following were declared unanimously elected:—

BOYD, JAS. A., 309, Smit Street, Johannesburg: Assayer. (Transfer from Associate Roll.)
CHAPMAN, J., c/o Fraser & Chalmers (S.A.), Ltd., Johannesburg: Engineer.
DEACON, C. O., c/o Messrs. Lewis & Marks, Johannesburg: Mining Engineer.

GOLDING, E., Falcon Mines, Ltd., Rhodesia: Assayer.

JEPPE, C. W. B., City Deep, Ltd., Johannesburg: Mining Engineer.

LOXTON, H. G. S., P.O. Box 1411, Johannesburg: Mining Engineer.

NEUFELD, E., P.O. Box 73, Denver: Chemist-Engineer.

POOLE, S. E., Witwatersrand G.M. Co., Ltd., P.O. Box 1, Knights: Mine Captain.

RIDGE, H. M., 2, Great Winchester Street, London, E.C.2: Metallurgist and Mining Engineer.

SEAGER, J. A., P.O. Box 6740, Johannesburg: Engineer.

NEW ASSOCIATES.

It was announced that the following gentlemen had been admitted by the Council as Associates:—

KETHEL, R. J. B., 31, Glen Deep Mine: Assayer.

LEE, S. P., Gracedale, Park Lane, Johannesburg: Company Director.

SIMPSON, W., c/o Messrs. Cawse & Malcolm, Johannesburg: Mechanical Engineer.

And as a Student:—

GLOVER, A. W., South African Institute for Medical Research, Johannesburg: Apprentice.

The President was sorry to have to report that the Society's Secretary was away ill; he had been sent home that morning by the doctor.

GENERAL BUSINESS.

OBITUARY NOTICES.

The President said it was his sad duty to have to announce the loss by death of two Past Presidents which had taken place during the vacation, namely, that of Messrs. Charles B. Saner and Hans S. Meyer. They were two personal friends of his, had done a great deal of work for the Society, and he would ask members present, as a mark of respect, to rise to their feet. (All members present rose.)

PRESIDENTIAL ADDRESS.

The President announced that the next business before the meeting was the delivery

of his Presidential Address. He would ask Mr. C. J. Gray (Vice-President) to occupy the Presidential Chair in the meantime.

THE PRESIDENTIAL ADDRESS.

By G. A. WATERMEYER.

It is one of the privileges of a newly-elected President to inflict on the members an address to which they must submit and from which they have no redress. This Society commenced as a Chemical and Metallurgical Society some thirty years ago, later it embraced mining and agricultural problems in its discussions, and we have gone so far as to make financial and statistical pronouncements which have carried world-wide weight. Our members have given us papers on original investigation in all these departments. We are naturally proud of the merit which has accrued to our Society as a result, and the hope must be expressed that the high standard of the past will be maintained in the future. To the younger members of the Society I would suggest that they come forward with papers in connection with the work on which they are engaged, and strive for the honour of carrying off the gold medal for the year in the department which they represent.

Industrial Conditions.—During the past year there has been a very grave struggle between the industry on which most of us depend for our livelihood and the various Trades Unions combined in the Industrial Federation. The Great War gave rise to conditions which in the nature of things were entirely different from former experience and consequently totally abnormal. Though the battlefield was Europe chiefly its influence extended throughout the world. Every source of production was speeded up to supply the wants of those engaged in the struggle, and the contending nations were willing to pay anything reasonable or unreasonable to have these wants supplied. The consequence of this was that labour was so greatly in demand that its price went up enormously. Parallel with this rise in price of labour the price of commodities rose, and with it the cost of living. It is a very difficult matter to correlate these different factors which are so interdependent, and the protagonists on either side were prone to

attach excessive weight to the factors with which they were immediately concerned at the expense of other factors quite as essential but not recognised as such.

Abnormal conditions cannot exist indefinitely, and the return to normal with the least amount of friction must be the object aimed at by good statesmen.

To the labour man it appears that wages once rendered possible can always be maintained at that standard, and any attempt at reduction must be strongly resisted. To the business man it is evident that if the cost of production of an article exceeds the price at which it can be sold the business is no longer sound and cannot survive. The problem is to get the labour man to see the business man's point of view, and conversely the business man to see the labour man's point of view. In order to arrive at this desideratum it is necessary to collect a large amount of data which must be correlated by specialists in statistics and adjudicated upon by trained economists.

It does not follow that the best advocate is going to make the best judge. This has been exemplified by the nugatory results of the late round table conferences. Each side came represented by a number of men whose minds were already made up and could not take a judicial view of facts. The spirit of conciliation was wanting, so that the conference presided over by Mr. Justice Curlewis broke up without any decision having been arrived at.

It becomes more apparent that the more conferences there are the less actual peace is effected. It points to a certain amount of insincerity on the part of those conferring. If by any chance an agreement be concluded one cannot gauge how long it will be kept. There seems to be no finality in compacts. The industry has tried the shop steward system of settling minor disputes. Unfortunately this led to the abuse of power by those who had not the wisdom to exercise discreetly the power with which they were invested. The managers on the

mines were rendered powerless to exercise the functions to which they were appointed. Discipline became lax, and consequently general efficiency suffered. It is clear that want of discipline would cause the downfall of any institution, whether it be a state, an army, or a mine. The various Trade Unions suffered under the control of good speakers but erratic thinkers, with the disastrous results which the Reef experienced during the month of March. The extreme element were in the ascendant, and forced the hands of the saner element, so that a strike developed into a revolt against constituted authority.

The Government, represented by the Prime Minister, used every endeavour to anticipate and settle the strike, unfortunately without success. After the revolt was subdued it promised a Commission of Enquiry which became known as the Mining Industry Commission. Its constitution was well balanced in the extreme. The personnel consisted of a Judge of Appeal as Chairman, a Privy Councillor experienced in dealing with industrial questions, the Principal of the University of Cape Town, who is a very clear thinker, and the Government Mining Engineer, who was par excellence the technical representative.

The most pleasing results of the sittings of this Commission is that the employers as represented by the Chamber of Mines and the employes by their various Unions have been brought together, and that the representatives of the latter have recognised that the conciliation scheme put forward by the Commission is worthy of a trial and recommended its acceptance.

Technical Education.—It is worthy of note that there is a great dearth at the present time of candidates presenting themselves for academic training as Mining and Metallurgical Engineers. The Witwatersrand University is exceedingly well equipped for supplying the necessary education, but very few students come forward to avail themselves of the facilities provided at this institution. An enquiry should be made into the causes for the shunning of these professions. Incidentally it may be mentioned that there seems to be no discrimination in high places in the industry between the diplomaed Mining Engineer and the product of the Miners' Training Schools. Though the latter is taught the manipulation of the various departments of underground work

his educational attainments cannot be very high. The engineer on the other hand goes through a very thorough training in all branches of science and engineering before he finally specialises in Mining and Metallurgy. The course at the local University occupies four years for its completion. The first two years are common to all engineering students, and consist in a grounding of Mathematics pure and applied, Chemistry, Physics, Geology, and Mechanics, including geometrical drawing, carpentry and workshop practice.

In the third year more advanced stages in these subjects are taken with the addition of Mining and Surveying with Metallurgy.

The final year is absorbed by Mining, Mining Economics, Surveying, Metallurgy with Assaying, Economic Geology, Mechanics, Electrotechnics, and Economics.

It will be seen that this is a comprehensive course which is designed to thoroughly equip the young engineer to attack the problems which will confront him in his subsequent practice. It would be idle to imagine that his education is complete when he obtains his diploma. What is, however, claimed is that he is fit to profit by what he sees, and is in a position to take responsibility gradually, and ultimately become a thoroughly competent engineer. Having spent four years after matriculation in study and a considerable amount of money in fees and board during that period, the graduate has a right to expect that he will be fostered by the mining industry and assisted sympathetically at the outset in the struggle for recognition of his ability and attainments. By so doing the mining houses will profit by the guidance of competent technical men who will strive for the efficiency of their departments.

Our University claims that her alumni should have a preference, other things being equal, for positions on the mines which demand the exercise of that special knowledge which they have acquired during their studentship. The mining houses would recognise, if careful records were kept, that it is the college-trained man who will become a most valuable asset.

Let us trace the history of the graduate. After matriculation he comes up to the University for four years' training at considerable expense to his parents or guardians. These years are spent in steady toil to acquire the knowledge which is essential to

the proper exercise of the functions which may subsequently be required of him. He then if he is lucky applies for and obtains a billet as sampler, assistant surveyor, or some other junior staff position at the remuneration of £15 to £20 per month. His preference then depends upon his chance of impressing his superiors and obtaining their goodwill. In due course he takes his examinations under the Mines and Works Regulations for the Certificate of Competency in Assaying, Surveying, Mine Overseer, or Mine Manager, which do not present extraordinary difficulties to him.

On the other hand a contemporary who may or may not have matriculated with him is in a position to apply to the mine office and obtain some junior position at a similar salary. He goes to some coach and is crammed with answers to questions which are likely to be asked in the examinations for Certificates of Competency, and is thus enabled to obtain a start of several years on the graduate. In many instances the graduate is dependent upon his educationally, ill-equipped contemporary for his promotion, and should there be a large number of such it is evident that through jealousy or other causes the ability of the graduate may remain unreported in higher quarters, and he may have to wait for dead men's shoes.

What we ask for is encouragement and opportunity for the young graduate, so that he may succeed in his aspirations in a reasonable time before he is put on the shelf as a phthisis pensioner. We require recognition of our graduates by the mining houses as highly educated men fit to become the future technical advisers, whether it be as heads of departments, Managers, or Consulting Engineers to the financial heads.

Our professions demand a very high standard of education for the proper exercise of their functions, but the young men must be reminded that the possession of the equipment is not in itself sufficient for success. The tact with which this equipment is used is what counts. The results attained by its use are the measures employed by the public in gauging the category under which a professional man will be classed.

Miners' Phthisis.—It is gratifying to note that each annual report of the Chamber of Mines refers to the improved conditions prevailing underground regarding the silica particles in mine air. Good work is done from year to year by the dust sampling departments both of the Government and the

Chamber. It shows that no stone is being left unturned which may tend towards the improvement of working conditions. The Symposium which has been conducted by the Society on this subject has been an education in itself. So many points of view have been brought forward that it must undoubtedly assist in furthering our knowledge and removing the cause of the disease. It was a happy idea of Dr. Orenstein to suggest the Symposium. He introduced the discussion, and his explanations from time to time have been very useful in assisting us to understand the medical points brought forward. As he mentioned originally it is the duty of everyone to come forward with ideas on the chance that some of them at least will be of assistance in solving the intricate problems which confront us respecting phthisis and silicosis. The idea of the Symposium is mutual assistance, and though criticism is welcomed it should not be acrimonious in its tenour. The members will look forward with pleasure to the time when Dr. Orenstein will gather up the threads of the discussion and present them in some concrete form.

Mr. C. J. Gray having pointed out that it was not the custom of the Society to discuss the Presidential Address, said he was sure they would thank their President for having given it. He had briefly traversed fairly wide ground, and had, among other things, referred to the peace which was now reigning on the Rand. Members would agree that peace and goodwill were what they wanted. As he went on he had referred to the question of the training of mining engineers. If they were to continue to make use of the peace and efficiency which they could claim at the present time they must look to the trained technical men of the future to tackle the new problems which would have to be faced by mining and those other industries which were growing up in their midst.

The President had also spoken of the progress made in the great fight against miners' phthisis. It was the general opinion that progress was being made. He thought they could look forward to the time when their troubles in dealing with the problems of miners' phthisis would be behind them; that they would be troubles on which they could look back with the feeling that they had conquered them.

He was sure they would join with him in a hearty vote of thanks to the President for his address.

STATISTICS OF EXTREMELY DEPRECIATED CURRENCIES.

By R. A. LEHFELDT.

(Printed in *Journal*, August, 1922.)

The President said, before he called upon Mr. Henderson to deliver his lecture

on "The Steel Age," he would like to announce that Dr. Lehfeldt had asked him to postpone the presentation of his paper until the subsequent meeting, as, unfortunately, he was unable to be present that evening, and preferred to make the explanations himself

THE STEEL AGE.

By J. HENDERSON.

The progress of mankind from savagery to civilisation has been popularly divided into the three Ages, Stone, Bronze, and Iron. That Stone came first is unquestioned, but whether Bronze preceded Iron or not is not quite so certain. There is no doubt as to the birth of the Steel Age, for it commenced with the reading of Bessemer's now historic paper at the Cheltenham meeting of the British Association in 1856. Before going any further it will be advisable to say a few words as to what steel is. Prior to Bessemer's invention the following definitions would serve to distinguish the two materials, wrought iron and steel.

Wrought Iron.—Iron containing practically no carbon. soft, malleable, ductile, easily welded, breaks with a tough fibrous fracture, but cannot be hardened or tempered, and will not carry a cutting edge.

Steel.—Iron containing 0.75 to 1.5% carbon chemically combined, hard, not easily welded, comparatively brittle, breaks with a close-grained fracture, can be hardened and tempered and carries a cutting edge.

The introduction of structural steel following on Bessemer's invention brought into use a material which in its softer varieties could be covered by the above definition of wrought iron. The following is a list of the various materials classed as steel:—

Carbon, .05 to .10%, dead soft steel, used for smithing purposes, tin plate.

Carbon, .10 to .20%, soft steel, general structural shapes, ship and boiler plates.

Carbon, .20 to .30%, general structural shapes, joists, axles.

Carbon, .30 to .40%, ordinary steel castings, light rails.

Carbon, .40 to .55%, heavy rails, tyres.

Carbon, .60 to .70%, spring steel.

Carbon, .70 to .80%, stamping dies.

Carbon, .90%, cold setts. This marks the boundary line between the tough steels and the hard cutting steels. .90% carbon is toughest steel that will carry a cutting edge.

Carbon, 1.00%, table knives and cold chisels.

Carbon, 1.10%, drills and large files.

Carbon, 1.35%, saw files.

Carbon, 1.50%, lancets and razors.

This list shows the absurdity of naming the softer varieties steel, as they are so entirely different in mechanical properties from the harder varieties. The term has, however, become fixed, and will probably remain so. In later years other kinds of steel have been introduced known as alloy steels, containing manganese, nickel, tungsten, vanadium, chromium, and molybdenum.

We have no record of when steel was first discovered, but its manufacture was not placed on a definite footing until the introduction of the cementation process early in the 18th Century. This process depends on the fact that if a piece of soft, carbonless iron is heated in contact with charcoal, but air excluded, the iron will absorb carbon from the charcoal which becomes chemically combined with the iron. The resulting product is entirely changed in structure and mechanical properties. The converted bars were made into piles, heated to welding heat, and hammered into bars and were known as shear steel. However carefully made, these bars were liable to be irregular in composition, and this lack of homogeneity brought about the introduction of crucible steel by Huntsman in the latter part of the 18th Century. He conceived the idea that if the bars could be melted and cast into ingots a homogeneous product would result. After many experiments he succeeded, and crucible

steel came into existence. With the exception of the use of manganese introduced by Heath about 1830, the manufacture of crucible steel is carried out in Sheffield in much the same way as in Huntsman's time. It is still the method used for the finer varieties of cutting steels. Before Bessemer's invention engineers had to rely chiefly on wrought iron for structural purposes. Bessemer was an engineer interested in gunnery, and was started on the path which ultimately led to his famous invention by the search for a stronger material for his gun experiments. He knew that in the manufacture of wrought iron from cast or pig iron that oxygen played an important part in the elimination of the impurities of the cast iron, so he first tried the effect of air blown through molten cast iron in a crucible. Then he tried the effect of blowing air through the molten cast iron, relying on the combustion of the impurities to supply the necessary heat to keep the whole molten without the aid of outside heat. The result exceeded expectations, for not only did the combustion of the silicon and carbon of the cast iron provide sufficient heat, but the metal at the end of the operation was actually hotter than at the start. Silicon, one of the chief constituents of cast iron, gives off on combustion to silica 7,830 Centigrade heat units or 14,094 B.T.U., and has this advantage over carbon in that the products of combustion being solid, the heat is retained in the converter and not carried away in the escaping gases. At this stage Bessemer showed his experiments to a friend who persuaded him to publish the results, which he did in a paper before the British Association at the Cheltenham meeting in 1856. The title of the paper was "The Manufacture of Malleable Iron without Fuel." The paper created only a mild sensation, and was received by the trade generally rather with incredulity and a certain amount of ridicule, while the British Association thought so little of it that they did not include it in the copy of their proceedings. Nothing daunted, Bessemer and several friends established works at Sheffield to develop the process on a practical scale. Right at the outset Bessemer struck two very serious difficulties. In the puddling process for making wrought iron the phosphorus in the cast iron is eliminated, and Bessemer quite expected the same would be the case in his process, but found there was no elimination of phosphorus at all. The reason is simple. In the puddling process

the slags are basic, rich in oxide of iron, which easily carries off the phosphorus as phosphate of iron. In Bessemer's process there is an excess of silica produced by the oxidation of the silicon of the cast iron, and also derived from the lining of the converter, which was composed of ganister containing over 90% silica. Thus any base present in the converter charge was all absorbed by the excess of silica, and so no base was available for the absorption of the phosphorus. Fortunately for Bessemer's hopes there was in England, in the Cumberland district, a very fine deposit of hematite ore sufficiently low in phosphorus to produce a pig iron suitable for his process. Thus this difficulty was overcome. Another difficulty was much more serious, and at one time looked as if it would wreck the process. After a charge is completely blown in the converter the resulting "blown" metal is practically pure iron, and naturally it was expected it would roll as easily as wrought iron; but when the ingots were put through the rolls they cracked and crumbled and behaved exactly like burnt or overheated iron. The cause of the trouble was oxide of iron, probably in a finely divided state, throughout the mass of the ingot. In the midst of their troubles and anxieties over this problem they received most encouraging news from Sweden, where with Swedish pig iron high in manganese excellent results were being obtained. Finally Mushet patented the addition of manganese to the molten "blown" charge, and this solved the difficulty. Manganese having a strong affinity for oxygen, carried off the oxygen combined with the iron, and passed off in the slag, and so rid the metal of its red-shortness. The manganese had still a further effect, as it also combined with the sulphur present. Sulphur as sulphide of iron has also a red-short effect on the material, but sulphide of manganese has not. There was a third benefit from the addition of manganese, for by adding it in the form of spiegeleisen, a cast iron containing 10 to 30% manganese and $5\frac{1}{2}\%$ carbon sufficient carbon was added to produce the various grades of structural steel required.

Thus was structural steel born, and it is impossible to estimate the enormous benefit it has conferred on engineering and world communications. In steel rails alone there was quite a revolution in railway progress and development. It would be impossible to run the heavy locomotives of to-day on iron rails, as they would prove too soft to

stand the severe demands of modern railway requirements. Not only could they be made cheaper than iron rails, but the output of a single Bessemer converter as compared with the old puddling process was enormous. In the puddling process it took $1\frac{1}{2}$ hours to make $4\frac{1}{2}$ cwt. of wrought iron, whereas the Bessemer converter turned out 10 tons of steel ingots in 20 minutes. In America a pair of converters produced the record output of 2,000 short tons of ingots in 24 hours. The Bessemer process has seen its best days, however, and must give way to the more reliable open hearth steel process. It still survives in the use of small converters for the manufacture of steel castings and also as a preliminary purifier in what is known as the duplex process.

The next development was the invention of the Siemens regenerative steel furnace. With this furnace Martin, in France, developed a method of steel making which was known as the Siemens Martin open hearth process. It was called open hearth because by means of doors in the sides of the furnace the charge was accessible for inspection and sampling. Martin melted pig iron and scrap on the hearth of the Siemens furnace, and added pure rich iron ore to refine the charge. Being accessible to inspection and sampling, the charge could be watched throughout the refining process, and by rapid chemical analysis was under perfect control. So reliable was the product that Lloyds adopted the Siemens open hearth process for all structural steel for shipbuilding, such as ship plates, boiler plates, beams, angles, etc. As the hearth of the furnace was made of silica sand there was no elimination of phosphorus, and therefore pig iron low in that element had to be used, and also the quality of the steel scrap had to be known. Naturally iron and steel metallurgists turned their attention to the problem of de-phosphorisation in the Bessemer converter. In 1878 Thomas and Gilchrist, two Royal School of Mines men, patented hard burnt dolomite as a lining for Bessemer converters. This material mixed with tar and rammed in position with hot rammers solved the problem as regards a suitable lining of basic character. Messrs. Bolchow, Vaughan & Co. had just completed a new Bessemer plant at Eston, near Middlesboro, and under the management of Mr. Windsor Richards undertook the experiments. Middlesboro pig iron containing $1\frac{1}{2}\%$ phosphorus was used. The converter was lined with the

hard burnt dolomite mixture, lime was added to the converter charge, and the blowing conducted exactly as in the ordinary Bessemer process. As soon as the carbon had gone blowing was stopped, and all expected to find the phosphorus eliminated. To their surprise and dismay practically all the phosphorus remained in the steel. They tried again and again, and nearly gave up in despair, when Mr. J. E. Stead (now the immediate past President of the Iron and Steel Institute) suggested going on blowing longer. No Bessemer man would have done this because it had become a fixed rule in Bessemer practice that the blowing must be stopped as soon as the carbon had gone, otherwise the iron would be wasted and so charged with oxide as to be past redemption. The suggestion was tried, and in the extra two to three minutes out came the phosphorus, and the problem was solved. Then it was found that it was as easy to eliminate $2\frac{1}{2}\%$ phosphorus as $1\frac{1}{2}\%$, and that the higher phosphorus was necessary to take the place of silicon as a heat producer. The slag produced in the converter when using pig iron with $2\frac{1}{2}\%$ phosphorus contained 18% phosphoric acid as phosphate of lime. One would scarcely expect that a phosphate of lime which had been submitted to a temperature of $1,600^{\circ}$ C. or $2,914^{\circ}$ F. would be any use as a fertiliser, but it proved on merely grinding to a fine powder and applied to the land without any other preparation to be a useful manure and so became a valuable bye-product. Its composition is a tetra basic phosphate of lime. The successful solution of the de-phosphorisation problem in the converter had a wonderful result as regards the development of Germany's iron resources. As that country had no ores low enough in phosphorus for the ordinary Bessemer process (henceforth known as the acid process) and was too far away from low phosphorus ore deposits in other countries, she was practically cut off from the benefits of the Bessemer process, but as soon as the basic process proved a success she hastened to take advantage of it. By this process she was able to develop her resources, and did so to such an extent that Great Britain was soon overhauled in steel output and left well behind. Germany took second place in the world's output of steel, America being first and Great Britain third. Even in 1913 when the open hearth basic process had made such rapid strides, Germany still relied on the basic

Bessemer for the greater part of her structural steel output. Her total output in that year was 18,355,000: basic Bessemer, 10,600,000; basic open hearth, 7,330,000; acid Bessemer, 155,000; acid open hearth, 270,000.

The successful solution of making steel from pig iron high in phosphorus advanced the manufacture of structural steel considerably, but there was urgent need for still further development. Iron ores sufficiently low in phosphorus for the acid Bessemer and acid open hearth processes are not very plentiful. Again ores sufficiently high in that element for the basic Bessemer process are also not extensive. But all over the world there are huge deposits too high in phosphorus for the acid processes and too low for the basic Bessemer. To utilise these for steel making required some kind of basic process which was eventually found by making the open hearth a basic one. The furnace is similar to that used for the acid open hearth, except that the hearth is made of hard burnt dolomite and magnesite instead of sand, and lime is added to the charge to form a basic slag. This process has now been so perfected that it has become easily the premier process in the manufacture of structural steel. It is under the same control as the acid open hearth process, and can in reason utilise almost any kind of pig iron and scrap. Lloyds, who would not look at basic Bessemer steel, and for a long time debarred basic open hearth, now accept the latter steel for ship building requirements. But for the successful establishment of the basic open hearth process structural steel could not have been manufactured in Australia, as the ores of that country are not suited for any other process. South Africa will also have to depend largely on this process because her ores are in the majority of cases similar to the Australian. In 1905 Great Britain made $3\frac{1}{2}$ million tons of acid open hearth steel and about $\frac{3}{4}$ million of basic open hearth steel. In 1913 the figures were: acid open hearth, 3,500,000 tons; basic, 2,800,000 tons. Since then the latter has grown further, all the new steel works being practically basic open hearth. In America, in 1905, basic open hearth steel was 7 millions, in 1913 21 millions. In Canada during the war 60% of the shell steel produced was basic open hearth. Naturally this is only a brief summary of such a huge subject, but before closing reference should be made to the wonderful assistance science has rendered in

the development of the industry. For years it was carried on without any scientific knowledge of what was being done, but now the successful manufacture of structural steel could not be accomplished unless carried out on strictly scientific lines. Steel is made daily in large quantities of any desired composition with a confidence that is marvellous when the difference between the raw materials and the product aimed at is considered.

Now that the South African Government has passed a Bounty Bill to assist in the establishment of the iron and steel industry it is to be hoped that those responsible will be up and doing, and establish the industry as quickly as possible. We can look forward to the time when all South African requirements of structural steel will be made here, and the surplus sent to less favoured countries. We can also look forward to the time when before this Society we shall be having papers on iron and steel problems in general and in South Africa in particular.

Mr. F. Wartenweiler in moving a vote of thanks to Mr. Henderson, thought he voiced the sentiments of the members of the Society in expressing his appreciation for the very able lecture that evening, also for the author's previous lecture. Personally, he had listened with great interest, not only because the lecturer had reviewed the subject historically, but also because he spoke with authority, he having made the working of iron and steel a life study and being practically engaged in it.

He would like to move a hearty vote of thanks to Mr. Henderson for his very able and interesting lecture.

Mr. F. W. Watson had very much pleasure in seconding the vote of thanks to Mr. Henderson for his very interesting lecture. He thought, as far as the mines at present were concerned, they were chiefly interested in drill steel. He might say that a lot of work had been done on this subject about 11 or 12 years ago, but matters had got to such a high pitch now that they could ask for a specification within narrow limits. This usually runs from .75 to .80 of carbon, phosphorus and sulphur together under .04, and manganese about .35. This showed the high pitch to which the steel industry had been brought, and had meant a tremendous saving to the gold mining industry. He was not speaking of structural steel, but drill steel, and the

improved quality owing to specification had saved thousands of pounds to the mines. To the best of his knowledge a better quality was being supplied at a less price than formerly was paid for an inferior article. He had very much pleasure in seconding the vote of thanks.

SYMPOSIUM: MINERS' PHTHISIS.

Mr. J. D. Marquard: During the twenty years I have been on these mining fields I have seen or known of comparatively few persons who have developed silicosis in crusher stations, sample crusher houses, or assay offices.

Turn with me underground, and names of miners can be given who developed silicosis in one year, at raising and winzing. I knew many developers who were quickly affected. Skipmen working in up-cast shafts have been known to get silicosis in comparatively short periods. Gangers have been known to quickly develop the disease in back stopes and other similar close places. Now why are persons more quickly affected underground than on the surface by silicious dust? My answer is mainly for three reasons, which are:—

- (a) The silicious dust is finer underground.
- (b) The silicious dust is more concentrated underground.
- (c) The atmospheric conditions are polluted underground, and such atmosphere aids the dust to get into the lung tissues.

Much has been said on (a) and (b); to-night I wish to touch on (c).

We are told that the lungs consist of cells to which the blood comes there to get into contact with oxygen from the air, and there to exchange for this oxygen carbon dioxide. Now we know that in this very room the air has silicious as well as vegetable and other mineral particles, and as we breathe this air and expel carbon dioxide most of these particles are expelled with the carbon dioxide. In other words, healthy lungs do not absorb, but have the power in good atmosphere to expel, dust and other impurities.

It is just this power which helped crusher men on the surface to keep dust out of their lungs, and it must be remembered that on the surface fine dust is also produced, especially in sample crusher rooms. If we turn to the animal world we find that horses

and cattle, in the dry season when travelling in dusty roads or grazing in dry fields, draw fine dust towards their lungs, but their lungs have the power of expelling such dust. Our lung cells, which have been designed to do their work under normal atmospheric conditions, it seems, refuse to do that work properly when working under abnormal atmospheric conditions such as we find in many of our mines. Such conditions are:—

- (a) Fumes from blasting.
- (b) Varying atmospheric temperatures.
- (c) High humidities.
- (d) Other impurities and deleterious conditions.

To me it seems under such conditions the lung cells get irritated or agitated underground, and in that irritated condition they have to absorb oxygen from impure air, expel carbonic dioxide gas, and repel dust.

To get the requisite amount of oxygen in bad atmosphere the lungs have to respire faster. On the surface we do about 18 respirations per minute, underground it might rise to 21 respirations when one is not doing excessive work, but think of a long steep back stope with foul air in it, think of the lung energy expended in walking up that stope, and it can be seen that often breathing will take place with as much as 36 respirations per minute, the same can be said of long ladder climbing in upcast incline shafts. We have seen, if the lungs are working under normal conditions, that they can repel dust; they evidently have been designed to do so. The velocity with which dust particles approach the lungs working under normal conditions is a velocity the result of 18 respirations per minute, but take a lung working in bad atmosphere with 36 respirations per minute, this naturally irritates the lung cells, and, on top of that, the dust particles come to those irritated cells with a high velocity, added to this is increased dust particles per minute owing to the higher velocity, so what can we expect: only that such lungs have not the power to repel dust particles or work properly against that velocity. It therefore seems to me that the velocity causes the particles to get well up against the lung cells in the same way as a moist sponge hung out to dry in a dust storm gets partially covered with dust. Konimeter work is an illustration of the effect of velocity with which dust particles travel; if the speed of air through the konimeter is below a certain velocity, then practically no dust is caught on the

vaseline. When carbon dioxide or carbon monoxide or nitrous fumes enter the lungs, besides irritating the lung cells, I ask medical members do these gases not act chemically on the lung fluids and temporarily form fluids of a new chemical composition; if so, are some of those fluids not of a sticky nature which arrest dust coming in at a high velocity. Does this arrested dust find its way into the lung there to form scar tissue, or is the dust expelled as the newly formed fluids are expelled. I mention this because with gassing cases one often notices sticky froth coming from the mouths of such persons.

Doubtless the fine dust is the chief cause of silicosis, but it appears that much of that fine dust which are about to enter men's lungs could be expelled by the said lungs if air conditions were brought to a higher standard underground. I believe our lungs have been designed to repel dust under normal conditions of lung action and atmosphere, but, in bad temperatures, the tendency is rather to absorb dust than repel it.

As an illustration that bad underground air conditions are detrimental to proper lung action, take two sets of mines, say the Geldenhuis Deep with the Jupiter, and the Rose Deep with the Simmer Deep. It is well known that in the deeper mines, the Jupiter and Simmer Deep, men developed silicosis sooner than in the Geldenhuis Deep or the Rose Deep. Was it because the quality or quantity of dust created by the different classes of mining work were different? No; machines in the 4 mines caused the same quality of dust, and blasting in the 4 mines gave the same kind of dust. Was the dust in the two deeper mines finer than on the other two mines. I do not think so. Why then did men get silicosis sooner in the deeper mines? Because the atmosphere in the deeper mines was more contaminated than in the shallower ones; this contamination was due to improper ventilation, the improper ventilation resulted in dust concentration, and the net result being polluted atmosphere with dust concentration acting upon lung cells weakened by increased respiration and irritation, and as can be seen from my former remarks under these conditions, the lungs were bound to absorb more dust in the deeper than the shallower mines.

Sufficient has been said to show that deleterious underground atmospheric conditions, other than dust contents, play a great part in the causation of silicosis. We have

room for improvement and we have to continue so fighting, but we have to take the other side as well, we have still to do a great deal towards improving the mine air. We have to bring the air conditions to such a point that the lungs will be able to aid us in repelling some of the dust which they now absorb. The main improvement we can look forward to for purifying the mine air is more ventilation. I do not mean putting up fans only and priding ourselves on the fact that we are pulling so much air through the mines; what is wanted is proper distribution of that air to the places where the mine air is contaminated such as in development ends, to such places where the lung cells get irritated and cannot repel dust because of bad air conditions.

Somehow I do not fancy ventilating dead ends with compressed air only. Normal atmosphere has certain living organisms in it. I ask the question from our medical friends, are those organisms necessary for healthy breathing, if so, do any of those organisms get killed under pressure of 60 to 80 pounds as we use in compressed air? How would we like to sit in this room day after day for 8 hours almost totally excluded from the outside atmosphere and the only ventilation we get is through an air pipe delivering air say at 60 pounds pressure. I think after a week our lungs will feel fairly groggy. Further, it is known that compressed air picks up lubricating oil in the compressor cylinders and also at the rock drills. This lubricating oil is exhausted into the mine air in a finely divided state. I ask our medical men, does some of this oil ever get to the lungs, and, if so, would it in any way affect the lungs, or does it get into the lungs in a carbonised state and perhaps aid the lungs or have a reverse effect.

I also have an idea that a large number of persons who develop silicosis get dust into their lungs in large quantities at a time in successive stages. On this matter I have a very open mind, and likewise would like to have medical opinion on it. To illustrate what I mean, take a gang of natives under a miner; they go into a drive some hours after blasting to clean up the blast on night shift; after working about a hour one native partially collapses and is gassed. The miner then takes his gang to fresh air at the station and despatches the gassed native up to the hospital. After waiting some time he and his natives return to the drive face and they work there the remainder of the shift. Possibly the majority of them have a

slight headache all that shift; this is almost certain, that their respirations would be much above normal and their lung cells would be severely irritated, and under these conditions quite an appreciable amount of dust would probably enter their lungs during the shift. Now the point is, if that miner worked in similar conditions many nights for a period of say 12 months he would probably be in the primary stage of silicosis at the end of these 12 months. Had this same miner worked in another drive for the same period where the dust contents in the air were the same as in the former drive but the atmosphere in the air good and cool, then I think he would not have got to the primary stage by the end of 12 months. If my idea is right, then it goes to show that gassy conditions prevent the lungs from repelling dust and really aid them to absorb it.

To summarise, suppose we had a lung here, and that lung could talk, and we said to the lung, "Well, lung, what is your opinion regarding the efforts of scientists and mining men to reduce miners' phthisis?" I think the lung would reply: "Gentlemen, I am very thankful for those efforts thus far in trying to ameliorate the amount of dust present in the atmosphere underground. I am sorry, though, that you have not taken me more into your confidence. I can help you somewhat and you should allow me to do so. You have brought the dust down well, and if those concerned underground do their parts well and honestly, then I can almost cope with dust still being created underground. I can prevent it worrying me, provided you give me carte blanche to do that for which I was partly designed. I must not be hampered as I have been hitherto. I must be given good air to breathe in the mine. I am getting a little tired of compressed air only in development ends; I like good air from the surface as well. You must keep on with your dust experiments, especially with regard to fine dust, but you must experiment freely regarding other health conditions, chiefly in the line of more ventilation underground. When you have got into line or to the same plane as you have got with your dust experiments, then that line or plane could be carried forward, and experiments continued on both the dust and other air impurities, each side keeping in step and helping the other side. You know I am sure I will watch you closely; my little cells will rejoice the more and more you assist me. Above all, give me not what some

call ventilation, but give me real, true and honest ventilation, then troubles I have been burdened with will in time ipso facto disappear, and what is more, this good ventilation will pull out so much fine dust in the mine that my little cells will be wondering what has come over the mines."

Mr. R. A. H. Flugge-De Smidt said that he had not been able to prepare a paper for that evening on the subject, but there were one or two things he would like to bring forward.

A controversy has been raging for some time past regarding the dust producing qualities of the Leyner type of drill.

The Government Mines Department consider that the air that accompanies the water feed through the hollow steel is dust-laden as it emerges in the form of bubbles from the hole that is being drilled. Attempts have therefore been made to reduce the amount of air passing through the hollow steel, and Mr. J. H. Veasey of the Chamber of Mines came out to Modder Deep to carry out some tests in this connection.

It was resolved to carry out four sets of experiments—one set with the ordinary Leyner and the remaining three with various types of improvements.

Each set of experiments was carried out in a certain development end where as far as possible the conditions were regulated so as to be the same on each occasion.

A fixed position was selected in which to take all samples. It was on a level with a man's head and approximately equidistant from face and sides of the Winze, which was 10ft. wide.

Eight to nine sugar-tube samples were taken one after the other, the duration of each sample being about eight minutes. A pump was used necessitating 110 strokes for 1/3 cubic metre.

At 30 strokes a spot was taken with each circular Konimeter. At 40, 50 and 60 strokes a spot each on a Kotze slide and at 70 strokes another spot with each Circular Konimeter. In all seven spots were taken to correspond with each gravimetric sample, so that sampling was practically continuous over a period of an hour and twenty minutes.

An extra water-hose was used throughout while collaring. Collaring took place during each set of samples.

TABLE SHEWING RESULTS OF EXPERIMENTS.

	Date of Sampling	Kotze' Konimeter Fish-glye on Slides Acid Treated, Dark Background Illumi- nation Ch. of M. Count		Circular Konimeter Vaseline on Slide Light Background Illumination Ch. of Mines Count.		Circular Konimeter Vaseline on Slide Dark Background Illumination Modder Deep Count Magnifi- cation 150.		Sugar-tube.		REMARKS.
		No. of Spots.	Av. Part per c.c.	No. of Spots.	Av. Part per c.c.	No. of Spots.	Av. part per c.c.	No. of Samples.	Av. Mgs m ³	
Long PistonLey- ner with solid cylinder washer	30.8.22	27	270	18	243	12	275	9	0.4	Machine had been drilling some time previously under same conditions.
Ordinary Leyner	5.9.22	27	1,190	18	864	18	878	9	0.6	Machine had been drilling some time previously under same conditions.
Long PistonLey- ner with grooved front cylinder washer	12.9.22	24	254	16	234	16	255	8	* 1.2	Machine had been drilling some time previously under same conditions.
Long PistonLey- ner with leather ring in front cyl- inder washer	12.9.22	27	241	13	166	18	150	9	0.1	Front cylinder washer changed after above tests, Mach. allowed to drill 15 minutes before sampling re-started.

* High result due to extra hose not been properly used while collaring.

One thing stood out very plainly, and that was that each of the different Konimeters recorded the same order of merit for the four sets of experiments.

The improvements in the Leyner machine had the effect of reducing the amount of air passing through the steel. In the last two sets of experiments practically no air was observed to be passing through the steel. Steep down-holes were being drilled and the holes were being excellently sludged. By minimising the amount of air passing through the steel with the water feed a vast improvement could undoubtedly be effected.

Mr. H. Pirow asked whether **Mr. De Smidt** could give any details as to the length of the dead-end in which the experiments were carried out, and whether there was any auxiliary ventilation or not?

Mr. Flugge-De Smidt replied that, as he had just stated, they tried to get conditions exactly the same in all four experiments. A winze was selected. It was dipping at 12° and had advanced 60 feet from the end of a drive. An 11in. galvanised ventilation pipe delivered 600 cu. ft. a minute at the brow of the winze. A 1in. auxiliary pipe delivered compressed air at a rate of approximately 60 cu. ft. of air per minute at about 25 feet from the face. The only thing they had to do was to try and regulate the flow of air as near as they could for all four sets of experiments. The experiments were carried out in the order named; the miner had blasted in the meantime; therefore, if anything, the conditions might be slightly worse as they proceeded with the experiments, being further from the delivery of the auxiliary air pipe.

The President asked if Mr. De Smidt would pass round the graph which he had prepared of the data, for the inspection of members. (The graph was accordingly handed round.)

Mr. Flugge-De Smidt said it was only the graph of one set of experiments, just to compare the different Konimeter counts—the three different countings. Each little circle represented the actual count. It would be seen that three Kotze Konimeter counts corresponded to two of the circular Konimeter counts; in each case they had a black curve for the Kotze Konimeter count; the red represented the Chamber of Mines circular Konimeter count; and the green was the curve of his count.

Mr. C. J. Gray thought it might be of advantage to some members if Mr. De Smidt would be good enough to explain the difference between the light ground counts and the dark ground counts.

Mr. Flugge-De Smidt said, first of all, it was the general practice for some time to count with a light background—that was, just using ordinary transmitted light; the particles showed up as little spots on a light background. They used to use vaseline on the slide as an adhesive.

Then the dark ground illumination was brought in; that was a method of cutting off a certain amount of the direct light and illuminating the particles by indirect light, when they saw the particles illuminated like little stars on a dark background. The dark background counts were very much higher than those of the light background counts. Then it was discovered that they very often had—particularly with samples taken in a fog—a very high count of very, very fine particles; but it was discovered by treating the spot with hydrochloric acid a number of those particles were dissolved, and one could generally take them to be calcium sulphate. He might mention that the dark background count of those spots taken with the Kotze Konimeter had been treated with acid; and the high counts were still there, despite the fact that they had eliminated quite a number of the particles. It was very difficult to make any comparison unless they knew under exactly what conditions a spot had been counted, because there was still a large number of microscopes not standardised on

the Reef. The Government Inspector of Mines used one of one magnification, the Chamber of Mines one of another magnification, and another party one of another magnification; and, as he had said, it was very difficult to compare one man's counting with another unless they got similar conditions. He did not suppose either of those three were absolute counts.

GOLD PRECIPITATION BY ZINC DUST AND IN CONJUNCTION WITH THE DE-AERATION OF SOLUTION AT MODDERFONTEIN "B."

By S. NEWTON and L. L. FEWSTER.

(Printed in *Journal*, June, 1922.)

DISCUSSION.

Mr. H. A. White: This Society is much indebted to the authors for the full and clear description of recent practice in gold precipitation and of the advantages to be derived from removing the greater part of the dissolved oxygen from solution as a preliminary to this operation. It would add to the value of their paper if their reply made clear which of the two statements as to the amount of lead nitrate added is correct and the exact tonnage of solution precipitated per filter leaf: the statement is made that 38 tons per filter leaf is passed through the press, but calculation gives only half that amount from the other data.

An analysis of runs 18 to 22 inclusive yields the following interesting results:—

(1) Composition of wet gold slime:—

Moisture	...	40.0%	4,260lb.
Zinc	...	27.0%	2,875lb.
Gold	...	16.0%	1,703lb.
Silver	...	1.6%	170lb.
Lead, etc.	...	15.4%	1,637lb.
Total	...	100.0%	10,645lb.

Assuming the lead nitrate added was 24lb. per shift, a total of 2,702lb. of metallic lead is represented by no more than 1,637lb. in the precipitate, and this indicates the probability that more lead nitrate was added than the solution dissolved in the time given, a point requiring further investigation.

(2) Allocation of zinc consumption:—

Remained in Press ...	42.63%	2,875lb.
Dissolved direct ...	32.09%	2,164lb.
Oxide formed ...	12.60%	850lb.
Lead pptd. ...	7.67%	517lb.
Gold pptd. ...	4.18%	282lb.
Silver pptd. ...	0.83%	56lb.
Total ...	100.00%	6,744lb.

The oxygen remaining in the solution after subjection to a 22in. vacuum will be about 1mgm. per litre or 0.2lb. per ton of solution, but it is hardly likely that a saving can be made economically at this point. A reduction in cyanide strength might slightly decrease the zinc directly dissolved (with release of hydrogen), but no great margin is obvious in this direction. Possibly the amount remaining in the press might profitably be reduced, but a margin of safety is desirable: on the other hand, a considerable reduction might give a product suitable for direct smelting, which is hardly the case at present.

(3) Press product per 100 tons of solution treated:—

Zinc ...	2.77lb.
Lead ...	1.58lb.
Gold ...	1.64lb.
Silver ...	0.16lb.
Moisture ...	4.11lb.

Total ... 10.26lb. per 100 tons of solution.

It seems probable that zinc dust feed might safely be reduced to 4lb. per 100 tons against 6.5 lb. shown at Modder B., and that metallic lead at the rate of 1.6lb. per 100 tons could be used in place of 2.7lb. as at present.

All the above deductions of course are liable to the uncertainty arising from the indirect calculations involved, but may be useful in indicating possible, though evidently not great, improvements in present practice.

An examination of the very lucid diagrams given reveals the interesting fact that the value of the tails appears to be quite independent of the rate of flow, of the value of the heads or of the pressure developed in the presses. The obvious fact that other things being equal the rate of pressure increase must depend upon the rate of flow is not very apparent.

In view of the fact that the introduction of de-aeration has seriously modified the relation between zinc-shaving and zinc-dust

precipitation of gold, a series of experiments is being undertaken at Geduld on the substantial scale of 300 tons of solution per day, and some indication of the results so far attained might be of interest at this juncture.

An ordinary 24in. Johnson Press is utilised with a belt feed for zinc dust with cone and float similar to that described in the paper—a sub-level delivery of the added solution being arranged in accordance with kind suggestions from Brakpan. The only disadvantage obvious with this form of press is that any stoppage due to power or other cause results in high tails for some time: the fall in pressure shows the cause to be a dropping from the filter paper of deposited zinc bearing slime and the remedy of adding an extra 2lb. of zinc dust to the cone before re-starting is obvious and efficient. The Merrill Press automatically provides against this difficulty in addition to the other advantages noted by the authors. While the authors lay stress on the necessity for maintaining cyanide strength above 0.017% and Brakpan rely upon an alkalinity of higher than 0.024% NaHO, the experimental results indicate that 0.008% KCN is sufficient with an alkalinity of 0.012% NaHO to keep the tails down to 0.01 dwt. So far we have not got below 4lb. of zinc dust per 100 tons of solution nor 1lb. of metallic lead actually in solution before the press. The experiments appear to indicate that the most vitally important factor is the presence of sufficient lead in solution, and no reliance can be placed upon additions to the slime pulp, but that addition should be made to the separated solution before the clarifiers and preferably before the pump used for that elevation. Trial will be made of the substitution of some form of lead oxide (at about 90% lead) in place of the more expensive lead nitrate (63% lead only), but only very finely divided material would stand much chance of success—red lead added just before the brown vats was a total failure.

It must be emphasised that the addition of lead in the case of the zinc dust process is a vital necessity, and the preliminary trial with solutions which are treated fairly satisfactorily in the ordinary extractor boxes showed a hopelessly inadequate extraction averaging about 50% only. On the other hand, the necessary additions of lead nitrate at the right point simultaneously improved the working of the boxes to the point of redeeming the de-aeration process from the

moderate advantages only shown previously to a pronounced success.

As illustrating the effect of rate of flow upon pressure it may be stated that while a rate of 2.2 tons per square foot of filter surface, with zinc dust fed at the rate of 7lb. per 100 tons caused the pressure to rise to 45lb. in three days, a reduction in flow to 1.8 tons of solution per square foot per 24 hours; with zinc dust fed at the rate of 4lb. per 100 tons gives a pressure rise to 15lb. only in 15 days. This suggests that the addition of a further filter press would remove the difficulty experienced at Modder B. in a cheaper and more effective manner than the introduction of any expensive and otherwise useless diluent. In view of the importance of lead salts in this connection the following table has been calculated from the data to be found in Comey & Hahn's "Dictionary of Solubilities."

Solubility of lead salts possibly formed in working cyanide solutions calculated to pounds per 100 tons of water:—

Lead Sulphide	0.05lb. per 100 tons.
Lead Carbonate	0.23lb. per 100 tons.
Lead Thiosulphate . . .	3.97lb. per 100 tons.
Lead Oxide	4.85lb. per 100 tons.
Lead Sulphate	5.60lb. per 100 tons.
Lead Thiocyanate	57.70lb. per 100 tons.

The white precipitate formed upon the addition of lead nitrate to working solutions is not lead sulphate as frequently believed, as is obvious from a consideration of above table, and may be demonstrated by the addition of a slight excess of sulphuric acid, which causes its instant disappearance unless sufficient lead is present to precipitate the much more soluble sulphate.

The above solubilities refer to pure water and the presence of alkali in working solutions enables much larger quantities of lead to be held in solution in spite of the carbonate always present. From a few experiments made I find that alkalinity to the extent of 0.010% NaHO (or 0.007% CaO) increases the amount of lead held in solution to the extent of roughly 1lb. per 100 tons of solution, and proportionally for stronger solutions up to about 0.050% NaHO, after which the amount held up increases at a much greater rate than the alkalinity. It is evident that the importance attached to amount of alkalinity when using the zinc dust process is due to the increase in solubility of lead so closely connected therewith.

The determination of lead in working solutions may readily be performed by a colouri-

metric method using about half a cc. of a saturated solution of sodium sulphide to 200 cc. of solution to be tested. This gives a perfectly distinct brown colouration when half a pound of metallic lead is contained in 100 tons of solution, and much less if a slight excess of acetic acid is added.

Mr. B. J. Hastings: In view of the paper on Gold Precipitation by Zinc Dust by Messrs. S. Newton and L. L. Fewster, published in the June number of the *Journal*, possibly it may be of some interest to members to note a few points on the procedure at Brakpan Mines, this being one of the few mines on the Rand using zinc dust for purposes of gold precipitation since the commencement of milling operations in June, 1911. The precipitation plant at Brakpan Mines is equipped with three Rumsey three-throw pumps, delivering at a maximum rate of 43.7 tons of solution per pump per hour, at the present rate of ram speed, each delivering to a Merrill Press of 40 units. This solution tonnage is equivalent to 3.150 tons per day, though the solution actually precipitated varies with the tonnage milled, being normally about 1.25 tons of solution per ton of ore milled or 1 ton of solution per Merrill Press unit per hour. Since the introduction of the Crowe Vacuum Process of de-aeration of solutions, the zinc dust consumption has been reduced to about 30% of the amount used before the installation, varying at present between .04lb. and .05lb. per ton of ore treated. Experience at Brakpan indicates that with this feed of zinc dust, precipitation is equally effective with solutions having a range of strength of .009% KCN to .03% KCN, solutions on this mine lying within these limits, the average value of filtrates giving .03 dwts. per ton by assay. The zinc feed totals 70lbs. per press for the first 24 hours after starting up, after which 29lbs. per day is sufficient, assuming a tonnage of 43 tons of solution going through the press per hour. It appears necessary to maintain a high degree of alkalinity in the solutions, anything below .018 CaO having a tendency to increase filtrate values. The introduction of lead nitrate in steady and sufficient quantities also appears to exercise a strong influence in obtaining efficient precipitation, the consumption at Brakpan averaging .035lbs. per ton of ore treated over the past few months. Smelting operations have benefited in respect of cost and labour. Acid treatment of the zinc gold slime from the Merrill Press still

continues, the resulting slime being calcined and pot smelted, producing a bullion of just over 800 fine. This bullion would be considerably finer were it not for its lead contents. The calcined slime is of a higher grade, returning over 5oz. fine gold per lb. of slime, or containing roughly 35% of gold.

I am indebted to Mr. C. K. Pitt, Acting General Manager, and Mr. J. L. Willey, Consulting Metallurgist, for permission to use these notes.

The meeting then terminated.

Notices and Abstracts of Articles and Papers.

CHEMISTRY

RATE OF ABSORPTION OF POISONOUS AMOUNTS OF CARBON MONOXIDE BY THE BLOOD.*—The experiments described herein—eight in number—were arranged in two series, four being done whilst “resting,” and the remainder while “working.” They were all carried out in a wooden airtight chamber lined completely with lead-sheeting and having a capacity approximately of 9,875 litres (say 350 cub. feet). The method of execution was as follows:—

After securely closing the chamber, the author was seated for approximately five minutes in order to allow the rate of breathing to become normal and regular, and a mouth-piece connected by flexible corrugated-rubber tubing to a two-way tap in a Douglas bag was worn. Air was inhaled from the chamber through a one-way valve of the Rosling type, and exhaled through a similar valve and the flexible tube into the chamber again. The carbon monoxide was next released from the container, and was thoroughly mixed with the air in the chamber by means of a small electrically-driven fan for a period of about three minutes. The exhaled air was then allowed to pass into the Douglas bag for a period which was accurately measured by means of a stop-watch. It was found that three minutes—while at rest—generally sufficed to fill the bag. At the same time an assistant took a sample of the inspired air from close to the mouth-piece. When the bag was filled, the volume of expired air was measured after being well mixed, and a sample taken in sample tubes, usually about 300 to 400 cubic centimetres (18 to 24 cub. in.). The percentage of carbon monoxide in both the inspiratory and expiratory air was determined, and the difference between them gave the percentage of carbon monoxide absorbed by the blood. In the “resting” experiments a second sample was taken after being exposed to the carbon monoxide for a further period, generally twenty minutes

longer, and samples were obtained in the same manner.

The second series of experiments was carried out while “working.” The work consisted of pedalling on a Martin ergometer, which is a bicycle having a cast-iron back wheel of known weight and circumference. A band-brake of linen is fitted, the ends being led to spring-balances, which are adjustable in order to obtain any desired tension. Then if the circumferential speed be known, and the tension, as shown by the difference in reading of the spring-balances in lbs. be also known, the amount of work in foot-lbs. per minute can be calculated. The work done varied between 4,805 and 5,405 foot-lbs. per minute, being on the average slightly over 5,000 foot-lbs. The amount of work is perhaps rather higher than that which an average person would do, and is approximately equivalent to walking on a rough road underground at a rate of three miles per hour.

The results are shown in the annexed table and Appendix. It will be noticed that in the first series the results shown as Ib, IIb, IIIb, and IVb all show a less percentage of carbon monoxide in the inspiratory air than in those shown as Ia, IIa, IIIa and IVa. This is, of course, accounted for by the fact that the author and his assistant were absorbing carbon monoxide the whole time, and consequently the carbon monoxide in the air of the chamber was being lessened in proportion. The percentage absorbed in this series varies from 50 to 44.6 for the first five or six minutes, and is considerably less for the next twenty or thirty minutes, being in the case of Ib and IIb a decrease of 9 and 7 per cent. respectively. With regard to IIIb, the drop in the proportion absorbed is less, being only 4 per cent., and in IIb there is actually a slight rise of 0.4 per cent. This can only be explained by the fact that the author was breathing very slowly and rather more deeply during the latter part of the test, and in consequence the carbon monoxide had more time to pass through the epithelium and combine with the hemoglobin. Dr. Haldane found some years ago that absorption of carbon monoxide, when small quantities were present, gradually fell away till a point was finally reached at which absorption ceased, so that probably the explanation of a lower percentage being absorbed after a greater length of time is due to the fact that the blood had already absorbed a great deal of carbon monoxide. With regard to IIIb, the reason that more carbon monoxide was being absorbed in relation to Ib and IIb is probably that the percentage in the inspiratory air was less, and a longer time would be needed to reach the point where absorption ceases.

With regard to the second series, it will be noticed that the percentage absorbed is slightly less, varying between 32.3 and 43.7; but it must be remembered at the same time that the rate of breathing increased considerably, and in consequence a greater volume of air was breathed. It might be thought that the amount actually absorbed would be no greater, since the air was held in the lungs for a shorter length of time; but it will be seen that the actual amount taken up is fully thrice the amount absorbed while resting and breathing

* Abstract of Paper read June 20th before the Institution of Mining Engineers. The author is engaged at the Mining Research Laboratory of the Birmingham University.

Tabulated Results of Experiments.

No.	First Series.								
	(1) CO. in inspired air.	(2) CO. in expired air.	(3) CO. absorbed.	(4) Volume breathed in litres per min.	(5) Minutes CO. breathed.	(6) CO. inhaled, in litres per min.	(7) CO. absorbed in litres per min.	(8) Proportion absorbed.	(9) Respiration per min.
	Per cent.	Per cent.	Per cent.					Per cent.	
Ia.	0.114	0.058	0.056	12.47	6.0	0.0140	0.00700	50.0	17.0
Ib.	0.107	0.061	0.045	13.76	30.0	0.0150	0.00610	41.0	18.0
IIa.	0.084	0.042	0.042	9.80	6.0	0.0082	0.00400	48.7	17.2
IIb.	0.080	0.045	0.035	8.15	30.0	0.0065	0.00280	43.0	15.3
IIIa.	0.047	0.025	0.022	11.90	4.5	0.0056	0.00250	41.6	16.0
IIIb.	0.044	0.026	0.018	8.28	31.5	0.0036	0.00150	40.0	14.0
IVa.	0.020	0.011	0.009	7.78	4.5	0.0015	0.00070	46.0	12.0
IVb.	0.017	0.009	0.008	5.75	24.5	0.0009	0.00046	47.0	11.0
Second Series.									
I.	0.091	0.057	0.034	45.19	5.0	0.0410	0.01530	37.3	Resp. 28; pulse 112; 4,805 ft.-lbs per min.
II.	0.073	0.050	0.023	36.69	6.0	0.0260	0.00840	32.3	Resp. 22; pulse 129; 5,405 ft.-lbs per min.
III.	0.053	0.029	0.024	48.03	5.0	0.0254	0.01110	43.7	Resp. 25; 4,914 ft.-lbs. per min.
IV.	0.026	0.015	0.011	44.35	8.5	0.0115	0.00480	41.7	Resp. 32; 4,992 ft.-lbs. per min.

the same percentage of gas. In experiments III and IV of this series, the actual amount of carbon monoxide absorbed is four and seven times more respectively than for the corresponding experiments of the first series. From a series of experiments carried out by Dr. Haldane and Mr. J. Ivon Graham (not yet published) it appears that the carbon monoxide is mainly absorbed during ordinary breathing, from the more superficial air cells in the lungs, and that very little reaches the deeper air cells. With the deeper breathing during work, more reaches the deeper air cells, and consequently more is absorbed.

The two main conclusions which can be deduced from the results of these experiments are as follows:—

(1) For cases where no work is undertaken in atmospheres containing carbon monoxide up to 0.114 per cent., a maximum of 40 per cent. of the amount inhaled will be absorbed. If it were possible to keep the rate and volume of breathing exactly the same for each experiment—which, of course, is impossible, since the physiological fitness of the subject is never exactly the same at different times—it would probably be found that the proportions absorbed would be at first the same for all percentages of carbon monoxide. It may be noted here as an interesting point that directly the author placed the mouthpiece—of the usual Proto type—in position, breathing became deeper and slower. The volume breathed in his case would be approximately 9 litres (318 cub. feet) per minute, and instead of the proportion varying, as it does, between 50 and 45 per cent., it would be found to be steady at about 48 or 48.5 per cent.

(2) For cases where work is undertaken in the same circumstances as in (1) a less percentage of the amount inhaled will be absorbed—say a maximum of 45 per cent. The table shows that a less proportion of the carbon monoxide inhaled is absorbed; but, as already pointed out, the greatly increased rate of breathing, and consequently the volume breathed, far more than makes up for this. In fact, the actual amount absorbed is in two cases twice, and in the other two cases four and seven times respectively, more than that absorbed while resting, as already stated.

The experiments bring out the fact that in actual work underground it would be as well to allow for an absorption rate of four times that of a person at rest. Consequently, a man would have only one-fourth of the time which he would otherwise have at his disposal in which to escape to safety. As shown, the amount of air breathed obviously has a great bearing on the amount absorbed, and to a certain extent the breathing is governed—under the same conditions of work—by the physiological fitness of the subject. A quadrupled absorption rate is therefore only approximate, and may be more or less. Further experiments which the author hopes to undertake shortly will perhaps decide this point.

The conclusion to be drawn from these experiments is that no one should ever go into a suspected atmosphere unless accompanied by a small warm-blooded animal, which should be a linnet for preference. A mouse or bird will during rest absorb carbon monoxide at a rate approximately fifteen times faster than a man who is also resting. But it must be remembered that where this test is to be used the person

engaged in exploring—or in whatever work he may undertake—will be walking along carrying the bird in a cage, so that it is obvious he will be absorbing a larger amount of carbon monoxide than if resting. In consequence, the factor of safety will be reduced to about a fourth—that is, the time in which he can continue to work or retrace his steps after the bird has collapsed will be reduced to this extent. The amount of work done—5,000 foot-lbs. per minute—corresponds very roughly, as already pointed out, to a man walking about three miles per hour, so that when the bird does collapse, or falls off the perch, the explorer should return at once, as there is little time to lose. In addition, a man is probably rather more sensitive to carbon monoxide than is a mouse or some birds. While underground the explorer, therefore, should be very careful to keep the bird under observation practically the whole time and not merely look at it—say, every five minutes—or the amount of collapse may not be noticed, and the time lost may be fatal.

APPENDIX.

PHYSIOLOGICAL EFFECTS.

Series I.

Experiment No. 1.—Slight temporal headache after fifteen minutes, which lasted only a short time. On coming out of the chamber, a decided weakness of the knees was experienced and difficulty in walking straight. After an interval of one-and-a-half hours had elapsed a very severe headache came on—both frontal and occipital. This lasted until the next morning.

Experiment No. 2.—Slight temporal headache during the experiment. After one hour had elapsed, a severe occipital headache came on, but lasted only about two hours.

Experiments Nos. 3 and 4.—No physiological effects felt except for a very slight temporal headache during the experiment.

Series II.

No physiological effects were felt in any of the experiments except a very slight temporal headache in Experiments Nos. 1 and 2, which may have been imaginary. No after-effects of any sort were experienced.—A. P. VEALE, *The Iron and Coal Trades Review*, June 23, 1922, p. 936. (J.A.W.)

PHYSIOLOGICAL EFFECTS OF CARBON MONOXIDE.—Although the extremely poisonous nature of carbon monoxide is understood, there is little available information on the physiological effect of small percentages of carbon monoxide in otherwise respirable air. It is always a poison, whether in small or large proportion, and in the latter case is the most dangerous gas encountered in underground operations. Fortunately it rarely occurs other than from a mine fire.

Doctors R. R. Sayers and F. V. Meriweather, of the Bureau of Mines, and W. P. Yant, chemist, recently concluded experiments to determine the physiological effects of low concentrations of carbon monoxide under varying conditions. The result of their investigation indicates that exposure for six hours to two parts

of carbon monoxide in 10,000 caused a 16 to 20 per cent. saturation of the hemoglobin of the blood, resulting in "very mild subjective symptoms of carbon monoxide poisoning" at the end of the test and no noticeable effects after the test. An exposure to four parts in 10,000 caused 15 to 19 per cent. saturation at the end of one hour; 21 to 28 per cent. at the end of two hours, and moderate to marked after-effects. Under the conditions of strenuous exercise, the time for saturation to 14 to 16 per cent. was one hour in a mixture of two and one-half parts in 10,000 and resulted in moderate symptoms of carbon-monoxide poisoning with after-effects mild to moderate. At higher proportions of carbon monoxide up to four parts in 10,000, the poisoning and after-effects were slightly increased.

According to the investigators, high temperature and humidity, with a given concentration of carbon monoxide, cause more rapid absorption of the noxious gas than do normal conditions of temperature and humidity. It is of interest to note that none of the subjects experimented upon has shown any permanent deleterious effects from the exposure to carbon monoxide.—DOCTOR R. R. SAYERS AND OTHERS, *Engineering and Mining Journal-Press*, May 27, 1922, p. 920. (H.A.W.)

CORROSION OF IRON AND STEEL.—This important subject has been extensively studied, and the present state of opinion with a summary of much of the literature was presented to the North-east Coast Institution of Engineers and Shipbuilders (Newcastle-upon-Tyne). Attention is called to the great waste that such corrosion involves and to the several theories of causes that have been proposed. It appears that iron does not rust in water containing only oxygen, but the presence of some substance giving rise to ionized hydrogen is required. Under ordinary conditions the active substance is carbonic acid, but, of course, other acids may be present, especially those produced by the oxidation of the organic matters of soil. The air of built-up areas is almost always contaminated with sulphur acids, and iron rust in such places usually contains notable amounts of sulphates. Electric conditions have an important bearing on corrosion. A difference of potential of the area exposed to water will determine the development of oxidation at certain points. Dr. Friend has recently investigated the subject with much care and offers a somewhat new theory. The oxidation of iron in contact with liquid water is catalytically accelerated by the ferric hydroxide hydrosol being alternately reduced by contact with the iron and oxidized again by contact with atmospheric oxygen.

Iron, in any of its forms, is attacked by a weak solution of carbonic acid, forming ferrous carbonate, which in the presence of free oxygen is converted into ferric hydroxide, thus again liberating carbon dioxide, which reforms the acid condition with the water, capable of repeating the process. It is evident that the small amount of carbonic acid normally in water may carry on the oxidation continually. Some years ago, a process for water purification was based on this principle. Ferric hydroxide has a high absorptive power for organic matters, but is active only in the fresh colloidal

state. By agitating water with iron scrap, fairly free from oil and rust, a green turbidity appears which soon changes to red. If now the water is filtered, a great deal of the organic matter will be held by the ferric hydroxide, which, of course, will be retained on the filter.

It has also been claimed that if bright iron is immersed in water practically free from oxygen and carbon dioxide, a minute amount of colloidal iron will form which will immediately oxidize if the water is allowed to come in contact with air.

Rust commonly met with in steel structures consists chiefly of iron combined with oxygen and water, forming a ferric hydrate, and has usually the familiar reddish brown colour, but the true colour is often disguised by the presence of deposited dust and traces of paint.

The product of corrosion is bulky, porous and hygroscopic. These properties tend to accelerate rusting, as the corroding influences in the form of vapours or liquids are attracted by and readily pass through the existing layer. Moreover, iron rust is electro-negative to iron, and furnishes innumerable local Voltaic couples in the presence of an electrolyte, which accelerate corrosion. Here, again, it will be observed that the hygroscopic nature of the rust constitutes a favourable channel for the electrolytic action. Upon removing apparently dry flakes of ferric oxide from a structure, a layer of moisture is often found between them and the steel. Thus, it can be understood that when rusting has once commenced it will readily proceed. Dr. Andrews showed that 50 per cent. more rust was deposited the second year compared with the first. It is unfortunate that the oxide, which forms on the surface of steel or iron assists in the process of rusting while the oxide coatings of other less commonly used metals such as aluminium, lead and copper, form a layer which protects the metal below.

It is recorded that some rejected plates from the Britannia Bridge, left unprotected upon a wooden platform exposed to the wash and spray of the sea, at the end of two years were so corroded that they could be swept away with a broom. They had an original thickness of seven-sixteenths to three-fourths of an inch. Basing calculation on the amount of corrosion which had taken place in the steel of the Britannia Bridge which had been kept well coated and clean, Baker found that it would take about 1,200 years to corrode the plates entirely. Roughly, the plates, which are well looked after, will last 600 times as long as neglected plates. These facts are mentioned to show that so far as corrosion is concerned, the serviceable life of a structure or vessel can be extended enormously by careful attention to cleaning and recoating.—A. PICKWORTH, *Journal Franklin Institute*, June, 1922, p. 839. (J.A.W.)

PLATINUM CONDITIONS.—The enormous advance in the price of this metal in late years has proved very embarrassing to chemists and to several industries. This advance has unfortunately contributed to additional difficulty because it has diverted the metal to a use for which it is really not adapted, namely, jewellery. There is no reason to doubt that the practice of setting precious stones in platinum has been adopted because of the high cost of

the material, for it has a poor lustre and does not set off the stone as well as gold. George F. Kunz has recently reviewed the conditions of the platinum supply and uses, and some of the data that he sets forth are here noted, being taken from recent issues of the *Chemical News*. The industry is gradually emerging from the chaotic condition into which it was plunged by the war, and even the Russian sources are beginning to be active. A notable increase of production is also recorded in Colombia, the locality, by the way, in which the metal was first detected. The Colombia mines were actively exploited while the Russian mines were blocked, but American companies are now endeavouring to stabilise the South American sources. The price of the metal has fallen somewhat, though still very high. An increased demand for jewellery and dental work has arisen since the close of the war which tends to keep up the price. In 1920 the consumption of platinum in the United States was 141,041 troy ounces, of which 57 per cent. was taken by jewellers, 19 per cent. by electrical industries, 11 per cent. by dental industries, 10 per cent. by chemical operations, the remainder being distributed in minor lines.

Naturally active search has been made for new platinum deposits, but so far no great rewards have come. Kunz states that the outlook for some Alaska exploitations is rather encouraging. In Colombia, the principal deposits are in the Atrato and San Juan Rivers, but a third river is regarded as likely to yield a supply. The United States is about to pay Colombia a large sum as indemnity, and it is hoped that much of this will be used to develop some of the Colombia industries, especially the platinum deposits. Undoubtedly a marked fall in the cost of platinum will be a great advantage to chemists.—G. F. KUNZ, *Journal Franklin Institute*, June, 1922, p. 861. (J.A.W.)

METALLURGY

METHOD OF GOLD-BULLION ASSAY.—The customary procedure adopted in assaying bullion for gold has persisted for so long without appreciable modification that the question of an alternative method has been considered by few. This is due largely to the fact that current practice permits an estimation of both gold and silver in the bullion. Recently, however, Arthur Westwood aroused interest at a meeting of the Institute of Metals by describing the following novel procedure, which has given satisfactory results. In this the cupellation phase is omitted, and copper is used in place of silver as the inquarting element.

The amount of bullion taken for assay, according to the originator of the method, is five grains—a fact that draws attention to the system of weights and measures still in vogue in England, even in precision work such as assaying. To this is added copper in amount so that the total weight will be from two to two and one-fourth times the amount of gold present. A gas muffle-furnace is used, the muffle being replaced by five or six silica tubes, 1½ in. internal diameter and 3 ft. long; these project from the front of the furnace for half their lengths. The copper-bullion tests are

melted, each in a clay cup of a shape to fit the tube, and each tube taking ten cups. The tubes are connected at the back of the furnace with a small boiler; the front ends of the tubes are open to the atmosphere. A steady flow of steam is maintained during the melting and cooling of the tests.

Melting, provided a temperature slightly above the melting point of copper is maintained, is done in five minutes. The tests are then withdrawn towards the open ends of the tubes to cool. On removal, the beads are hammered and rolled in the usual manner. Annealing is unnecessary. The fillets are then folded, and parted with acid.

The new method has the merit of extreme simplicity, its adoption should result in an appreciable economy of time and labour in those operations where an estimate of silver content is not required. Mr. Westwood has published sundry data in support of its accuracy; but the scope and application of a non-cupellation method in general metallurgical work remains to be estimated. The fundamental idea is attractive and will doubtless appeal to many assayers.—A. W. ALLEN, *Engineering and Mining Journal-Press*, June 17, 1922, p. 1057. (H.A.W.)

State of Origin. Material.

Alabama	Quartzite
California	Ryalite
California	Olivine Basalt
California	Altered Basalt
California	Diorite... ..
California	Basalt
Connecticut	Diabase
Missouri	Chert

TESTING PEBBLES FOR TUBE MILLING.—Hard flint pebbles, usually gathered on the beaches of Denmark, formed the crushing medium in early tube milling, except on the Witwatersrand goldfield, where the banket ore supplies the pebbles. With greater demand for pebbles the quality deteriorated, and control of and expense of tube milling became a serious factor. Operators in the United States of America of necessity turned to the possibility of using local hard rock, and investigations were conducted which have resulted in practically displacing all imported pebbles.

Laboratory physical tests are carried out on the rock, and these have been found to give a practical index of the value of the stone for a grinding medium. The tests include determination of hardness, toughness and abrasion, and are conducted by methods and apparatus used for the determination of the value of road making material. These are the subject of publications issued by the Office of Public Roads and Rural Engineering, Washington.

Following are given natural pebble-making rocks with a toughness co-efficient of 18 or over:—

Wgt. per cub. ft. Resistce. to wear. Hardness.

165	11.7	19.7
165	23.0	18.7
175	24.1	18.9
181	27.4	19.2
184	23.8	18.7
181	19.6	18.2
187	21.1	18.3
125	5.8	19.6

—A. W. ALLEN, *Mining and Scientific Press*, March 25, 1922, p. 405. (F.W.)

MINING

THE RESISTANCE TO TRACTION OF MINE TUBS.*

—The forces opposing motion of a tub are: (a) Friction at the bearings of the tub; (b) friction between the wheels and the rails; (c) accelerating the tub in starting from rest; (d) gravity, which comes into play in hauling uphill; (e) impact of air currents.

Of these, (a) is a maximum on starting the movement; (b) is greatly dependent on the kind of wheel and rail and upon the curvature (if any) of the track.

Methods of Determining the Coefficient of Friction.—It is a long time now since Mr. Nicholas Wood† made his exhaustive enquiry into the resistance offered to traction by mine tubs. Since his paper was read, many tests‡ of this character have been carried out; but believing that good would be done by revising these experiments and thereby calling notice to

the need for improvement in our mine rolling-stock and underground railways, the author recently conducted a series of tests at the Heriot-Watt College, Edinburgh, and at a number of collieries. As a near approach to actual mine-working conditions, the gallery at the Mine Rescue Station lent itself admirably.

The two tubs, a wooden and a steel tub, were only in average condition. The wooden tub had a tare of about 3½ cwt., the diameters of the axle and the wheel were 1½ in. and 10 in. respectively, and the wheel-base 18 in. The empty steel tub weighed 4½ cwt.; the axle and the wheel were 2 in. and 12 in. in diameter respectively, and the wheel-base was 17 in. The rails were round-headed and flat-bottomed, weighing 18 lbs. per yard, and the gauge of the track was 2 ft. The bearings were ordinary hutch pedestals, with no special form of lubrication, the axles being fixed to the wheels. Other types of bearings were to hand, and could be fitted on for the purpose of comparison. The following methods of experiment were adopted—

(1) A fairly level portion of the track was selected. A spring-balance having a dial 15 in. in diameter, and reading to 150 lbs. was used to register the pull. The balance was attached to the empty tub, and a steady horizontal pull

* Abstract of Paper read, June 10, before the Mining Institute of Scotland.

† *Trans. N. E. Inst.*, 1854-1855, Vol. III, page 251.

‡ *Ibid.*, 1867, 1868, Vol. XVII, Appendix i, pages 87 and 131.

was given. Readings of the dial were taken at regular intervals till a distance of about 30 or 40 yards had been traversed. The spring-balance was then detached and hooked on to the other end of the tub, and readings of the dial were taken on the return journey. The readings were taken every two yards, the rate of travel being about one and a half miles per hour. Neglecting the high readings due to the effect of acceleration, the average of all the other readings was taken as the mean pull. The weight of the tub and its load was measured on a delicate steel-yard which had been recently tested. To note the effect, if any, of increased weight, the weight of the empty tub was increased by increments of 1 cwt. till a load of 10 cwt. was reached (Fig. 1).

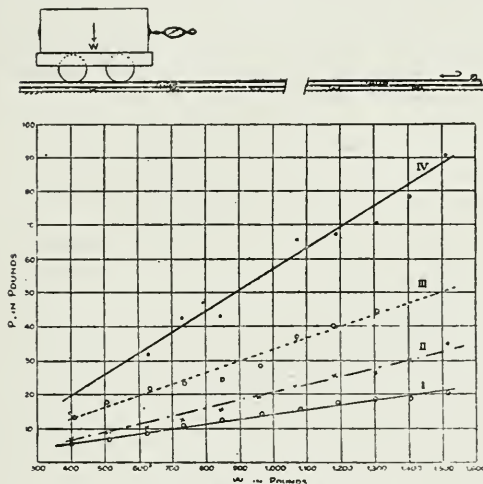


FIG. 1.—METHOD No. 1.

- I.—Good lubrication.
- II.—Average lubrication.
- III.—Poor lubrication.
- IV.—Starting effort with average lubrication.

(2) This was in principle the same as No. 1. Two pairs of rails were adjusted perfectly level and the tub placed thereon. A cord passed horizontally from the front end of the tub to a deflecting pulley, around which it was passed and up over another pulley at a height of about 14 ft. above floor-level. To this end of the cord was affixed a hook for the purpose of attaching and detaching the carrier containing the weights. As a preliminary, the weight necessary to overcome the friction of the cord and pulleys was ascertained and allowed for during subsequent tests. A slight assistance was given

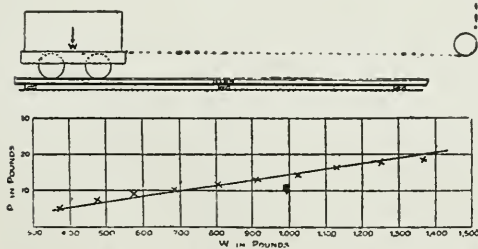


FIG. 2.—METHOD No. 2: GOOD LUBRICATION.

to the tub to overcome inertia, and the weight at the end of the cord which just kept it moving along without increase of speed was duly noted. Each test was repeated three times and the average weight was taken as the pull required (Fig. 2).

(3) If when a body is placed on an inclined plane, at such an angle from the horizontal that the force due to gravity ($W \sin \phi$), acting down the plane, is just balanced by the total resistance due to friction ($\mu W \cos \phi$), this angle is termed the angle of friction (W =weight, in lbs.; μ =coefficient of friction). In these circumstances, if the body is started from rest it continues to move at a uniform speed. The tangent of the angle of friction is equal to the coefficient of friction.

Method No. 3 consists in determining the angle of friction of the tub and thereby the coefficient of friction. In order to carry it out a pair of heavy rails weighing 25 lbs. to the yard were bound together by tie-rods (one at each end and one midway between) so as to maintain rigidity and the proper gauge. The tie-rod at one end was placed in pedestals, and at the other end of the rails was attached a flat wooden sleeper, under which a pair of slow-motion screw lifting jacks were placed. The track was first made level and a spirit-level fixed on the flat sleeper at the raising end to ensure that the lifting was done evenly and without canting of the track. The tub was then placed on the rails and the screw-jacks started, slowly raising one end of the track. Now and again a slight push was given to ascertain when the tub would just move off without gaining speed. When this point was arrived at, the inclination of the track was taken by a clinometer. To ensure accuracy, at least three trials were made, care being taken in each case that the tub was square on the track and not twisted with the fingers bearing hard against the sides of the rails. Neglect of this precaution greatly affected the results obtained. In this experiment, in addition to the foregoing, the angle necessary to overcome inertia and cause the tub to move off at its own accord (of course, at an increasing speed) was also determined (Fig. 3).

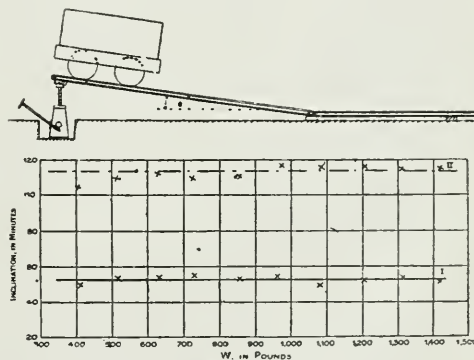


FIG. 3.—METHOD No. 3, WELL-LUBRICATED TUBS.

(4) A portion of the track at one end was elevated to allow the tub to run off under gravity and come to rest on the flat portion of the

track. The track was made straight with rails in true alignment, and care was taken to avoid any sudden change of gradient. The distance travelled, in feet, and the difference of level between the starting and stopping points of the tub were measured. The average of three tests was obtained. The difference of level, in feet, divided by the distance in feet travelled, was taken as a measure of the coefficient of friction (Fig. 4). In this case the effective work

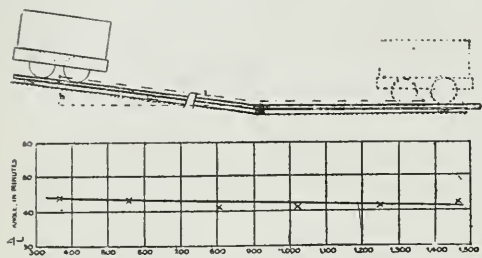


FIG. 4.—METHOD NO. 4: STARTING EFFORT;
GOOD LUBRICATION.

done by the descending tub ($W \times h$) is absorbed by the total resistance due to the friction ($F \times l$). W =weight in the h =difference of level, in feet, between starting and stopping points, F =friction in lbs.; l =length, in feet, between starting and stopping points.

As a result of the experiments, the author concluded that, given a properly calibrated spring-balance, Method No. 1 is the best. In addition to the pull necessary being got direct, any inequalities of gradient and wind pressure are neutralised by travelling the same part of the track in opposite directions. Method No. 2, although giving fair results, suffers from the drawback that the distance travelled is too short, and that therefore there is insufficient time to note if the motion is uniform; and, further, the sample of road taken is inadequate. Method No. 3 has the same objections, and the proper placing of the wheels on the rails at the starting point requires much attention. Though Method No. 4 is simple, and the results obtained were fairly good, it is not practicable as with Method No. 1 to repeat the observation in the reverse direction; and it was found that low results were forthcoming when the air current assisted the movement.

The fact that, of these methods, the only one which can pretend to determine the coefficient at specific rates of movement is the first, is a decided advantage of that method.

Types of Bearings.—In the case of the wooden tub at the Rescue Station, the only form of bearing used in the tests was the ordinary open variety so generally used with fast wheels. The steel tub, on the other hand, was tested (a) with the open type of bearing; (b) with the Climax wrought-iron bearing, in which there is a special form of lubricator; and (c) with Skefko ball-bearings, using the same axles. In the tests at collieries referred to below, the bearings were for the most part the plain or open variety; but at one pit opportunity was given of testing bearings on the Rowbotham principle (loose wheels).

It is to be regretted that no chance was afforded of testing roller-bearings, a form which has come much to the fore in America. According to figures supplied by makers of roller-bearings for tubs, the coefficient of friction is, in comparison with plain bearings, reduced at least 50 per cent., and the author sees no reason to doubt the accuracy of that statement.

Summary of Results for Straight Roads.—Figs. 1 to 4 set forth the results obtained in testing the wooden tub (plain bearings) at the Mine Rescue Station. Fig. 1 clearly shows the advantage accruing from good lubrication and also illustrates the large additional effort needed in starting motion. The result given by Method No. 2 (Fig. 2) agrees closely with that obtained for the well-lubricated tub by Method No. 3. Fig. 3 shows that, within the limits of experimental error, the angle of friction is substantially constant with varying loads; Fig. 4 indicates a slight reduction of that angle with increasing loads, but the variation is too small to be of practical importance. The graphs furnish the data recorded in Table I.

TABLE I.

Co-efficient of Friction (Wooden Tub, Plain Bearings.)

State of bearings.	Speed.	Co-efficient of friction.			
		Method No. 1.	Method No. 2.	Method No. 3.	Method No. 4.
Poor lubrication	About 1½ miles per hr. Starting from rest.	1:30	—	—	—
Average do.		1:54	—	—	—
Good do.		1:68	1:70	1:65	1:80
Average do.		1:20	—	—	—

The figures in Table II. relate to the steel tub at the Rescue Station, it being fitted as stated above, with bearings of different kinds.

TABLE II.

Co-efficient of Friction (Steel Tub).

Type of bearing.	Co-efficient of friction
Plain (open)	1:58
Climax	1:60
Skefko	1:135

Table III. includes the results obtained in testing tubs under working conditions at four mines. The coefficients stated are in each case the average of a number ascertained for different tubs. With the exception of the last instance, where the tubs held one ton of coal, the capacities were 8 cwt. to 10 cwt. Method No. 1 was adopted.

TABLE III.

Tests on Tubs at Four Collieries.

Type of bearing.	Diam. of axle.	Diam. of axle.	Character of rail.	Method of lubrication.	Co-efficient of friction.
Plain ...	Ins. 12 (edge-wheels)	Ins. 1½	Angle	Hand	1:20
Do. ...	9	1½	Bridge	Hand	1:33
Do. ...	10	1½	Bridge	Automatic greasers	1:35
Rowbotham	11½	1½	Round-headed	Round-pump	1:65

Increased Friction Due to Curves.—To determine the effect of this, a 9ft. radius curve of

90 deg. was constructed. This was laid down as level as possible and the gauge of the track made 1 in. wider than that of the straight road. It was found that the average effort necessary to take fast-wheel plainbearing tubs round this curve was over three times greater than on the straight (Fig. 5). This increase seems considerable, but it must be remembered that the curve is very sharp, and jamming and skidding of the wheels occurred to a very appreciable extent. With larger curves the extra effort is not so great; and in cases observed at two different collieries this proved to be the case. With fixed wheel and with a curve radius of 12 ft., the effort was there found to be about twice, and with loose wheels on a curve of the same radius, about one-and-a-half times the ordinary effort.

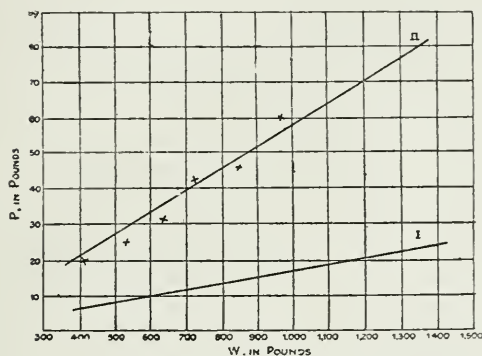


FIG. 5.—SHOWING INCREASED RESISTANCE DUE TO CURVE.

I.=Coefficient of friction for Straight Road.
II.=Coefficient of friction for Curve.

Starting Effort.—Fig. 1 and Table I. indicate the large extra effort needed in starting a tub from rest, and point to the necessity of providing haulage engines of a power ample to overcome this excess. The results indicate that with plain bearings the effort necessary to start a tub from rest and to accelerate it to a uniform speed of $1\frac{1}{2}$ miles per hour required two to three times the effort necessary to maintain that speed. With the Rowbotham type of bearing the equivalent starting effort was found to be one and a half to two; and with ball-bearings from one to one and a half times the ordinary effort.

Friction between Wheel and Rail.—The tractive force necessary to overcome the resistance encountered by the wheels rolling on the rails is included in the foregoing results, but in straight roads it is small compared with the friction of the axles and bearings. In order to determine its amount, a pair of wheels were detached from one of the tubs and carefully placed on a level piece of track. One end of the track was elevated sufficient to cause the wheel to move off without increase of speed. The inclination to accomplish this was found to be 1 in 363, a figure which thus represents the coefficient of friction of the wheels on the rails on a straight road. With uneven, rough, and dirty surfaces and on curves this coefficient will be greatly increased.

Coefficient of Friction of the Bearings.—In order to ascertain the friction at the journals independent of any other cause, the axle was placed in the bearing with the hutch upside down; a string wound round the axle enabled a measured force to be applied at the same leverage as the force of friction at the bearings. The weight of the axle and wheels having been ascertained, it was found that the pull in the string amounted to from 1-6th to 1-11th of the weight resting on the bearings. The axle and bearing contacts were not perfectly smooth and in only average state of lubrication and cleanliness. With ball-bearings, this coefficient was found to be 1-50th. Actually, the force causing the wheels to rotate is applied at the radius of the wheel; hence if the wheel has six times the radius of the axle, the friction coefficient (bearings only) will amount to a sixth of the figures just stated. It is thus apparent that the coefficient of friction of a tub with a given size of axle is an inverse ratio to the size of the wheel.

Sliding Friction.—In order to test the effect of spragging the wheels, the steel tub with fast wheels was used on a straight piece of track. The following results were obtained: (1) With back wheels alone spragged, the inclination at which the tub would just move downward without increase of speed was found to be 1 in 6. (2) With front wheels alone spragged, this inclination was 1 in 4.8. (3) With both back and front wheels spragged, the inclination was 1 in 3.3.

Resistance of Air Currents.—For ordinary purposes we might consider the resistance encountered by the passage of tubs along roads in which air travels at ordinary velocities to affect the ventilation more than the haulage. However, in a test conducted on a level piece of track in an air current of about 300 feet per minute, there was observed a difference of about $\frac{1}{2}$ lb. in the tractive force necessary to pull the tub in the two directions. With higher velocities of, say, 20 to 40 feet per second, this difference would be much augmented; but even this is barely worth considering, having regard to the much greater resistances of other descriptions that have to be overcome.

Speed of Haulage.—According to experiments that have been made in America by Mr. Van Norris,* the higher the uniform speed of haulage the less is the tractive effort required; but in tests conducted by the present author the opposite effect was apparent. With a tub having a gross weight of 900 lbs., the effort to maintain a speed of 0.85 mile per hour was 13½ lbs., and at a uniform speed of 13 miles per hour an effort of 16 lbs. was required to keep the same weight moving.

Lubrication.—The state of lubrication of the bearings has an obviously important effect on the coefficient of friction of a tub. With a plain-bearing tub giving on test a coefficient of friction of 1-30th, it was found that on the

* "Coal Miners' Pocket Book," page 165.

tub being well lubricated (for the time being) the coefficient was reduced to about 1-70th. With this type of bearing lubricated by hand, or by automatic greasers, there is an enormous waste of grease. The grease constantly dropping off en route tends to make the roadways filthy, and at the same time an inflammable material is deposited which increases the danger to the mine. Moreover, dust, grit, etc., are always getting on to the open bearing, thus increasing the friction and wear of the axle and bearing. On enquiry at a few collieries the author obtained the cost necessary to keep the tubs lubricated, as follows:—

(a) With grease at 11s. per cwt., the cost for grease worked out at 0.36d. per ton raised and on the number of hutches in use this is equal to 12s. per hutch per year. In this case the tubs had plain bearings and were hand-greased.

(b) At another colliery the cost for grease was £400 per annum, or 0.3d. per ton raised, and 9s. 3½d. per tub per year. In this case the tubs had also plain bearings and were lubricated by automatic greasers.

(c) From results obtained at one colliery in Warwickshire, where fast wheels were used, the friction coefficient was found to be 1-32, and some years before the war the grease cost was 2s. 5d. per tub per year.

(d) At a neighbouring pit, where loose wheels were used, the coefficient was 1-550, and the grease cost 1s. per tub per year. The tubs with loose wheels had twice the mileage of the others.

A maker of bushed bearings (loose wheels) states that the cost for lubrication at a certain colliery was reduced from £400 to £60 per year by changing from plain bearings to bushed bearings running in oil.

A maker of roller-bearings states that with that particular type of bearing the cost for lubricant is only 8d. per tub per annum at the present day.

The first two cases (a) and (b) are very striking in showing the amount of needless cost and waste in the lubrication of plain bearings.

Requirements of Modern Tub Bearings.—The essential requirements of a good type of tub bearings are: (1) The bearing should have the minimum of friction resistance. Siding friction (plain bearings) should be converted into rolling friction (roller or ball-bearings). (2) The bearings should have efficient and continuous lubrication, with an expenditure of the minimum of lubricant. (3) They should be immune from the entry of dust and grit. This and the last requirement involve the use of closed bearings. (4) The ratio of wheel diameter to axle diameter should be as high as practicable. (5) The bearings, axles, etc., should be strong enough to withstand ordinary rough usage. (6) They should be easy and cheap to repair and renew. (7) They should have a reasonably low first cost. (8) The tub should be designed to facilitate its passage round curves.

Some dubiety may exist as to the behaviour of roller or ball-bearings under the severity of underground use; but with axles, wheels, and bearings made of the strongest material

and of proper design, there should be little doubt on that score. Hundreds of thousands of heavy tubs of these kinds are in use in America, a sufficient testimony to their reliability.

CONCLUSION.

From the results obtained, it is obvious that the plain type of open bearing has little to recommend it unless on the score of simplicity and cheapness. It is a remarkable fact that after a lapse of well over fifty years in British mines we find, generally speaking, no improvement in the condition or character of tub bearings. The coefficients of friction ascertained by Wood were in many cases less than those found in the author's tests.

The saving in power for haulage and lubrication that may be effected by the adoption of a more modern type of tub will in almost every case be considerable. With a coefficient of friction about half and with the cost of lubrication only about one-tenth of that of the old type, the newer forms are surely worthy of consideration.

Along with an improved type of tub, the laying and maintenance of the underground tracks in many mines would require much greater attention. Badly-laid tracks on unstable foundations increase greatly the tractive force necessary, and by causing derailments augment damage to tubs, and tend to decrease of output. Heavier and more stable tracks will be required, and a tendency towards the general adoption of larger and stronger tubs should be encouraged; they would reduce materially the amount of rolling stock and the number of journeys necessary to deal with a given output. Attention to underground roads, somewhat akin to that paid by our railway companies to surface railways, would certainly bear good results. One cause of American mining efficiency is due to the special attention paid to mine transport.

A great amount of time and research has been spent in determining the efficiencies of power, pumping and ventilating installations, and there is little doubt that more attention ought to be paid in investigating the efficiency of one of the most important (if not the most important) installations of a mine, namely, the haulage.—JOHN WILSON, *The Iron and Coal Trades Review*, June 16, 1922, p. 881. (J.A.W.)

CHANGES OF ADDRESS.

ALLEN, W., *l/o* Crown Mines: Modder "B" Gold Mines, Ltd., P.O. Modderbee.

BALL, H. STANDISH, *l/o* London: Apex (Trinidad) Oilfields, Fyzabad, Trinidad, B.W.I.

COCKING, A. T., *l/o* Umbogintwini, Natal: c/o Kynochs, Ltd., Birmingham, England.

JAMES, J. H. D., *l/o* Con. Langlaagte Mines, Ltd.: Van Ryn Deep, Ltd., P.O. Box 225, Benoni.

LLOYD, J. J., *l/o* Madagascar: 117, Northdene Avenue, Brakpan.

LOWRY, W., P.O. Box 7003, Johannesburg.

MAIN, A. F., *l/o* New York: P.O. Box 109-Bis, Mexico City, Mexico.

THE JOURNAL

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Chemical, Metallurgical and Mining Society

OF SOUTH AFRICA

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Vol. XXIII.

OCTOBER, 1922.

No. 4.

Proceedings

AT

Ordinary General Meeting, 21st October, 1922.

The Ordinary General Meeting of the Society was held in the Assembly Hall, Scientific and Technical Club, 100, Fox Street, Johannesburg, on Saturday, the 21st October, 1922, at 8 p.m., Prof. G. A. Watermeyer (President) in the chair. There were also present:—

27 Members: Messrs. H. R. Adam, C. J. Gray, F. W. Watson, J. R. Thurlow, J. Chilton, R. A. Cooper, J. H. Johnson, H. Pirow, J. M. Thorburn, John Watson, A. Whitby, H. A. White, J. L. Willey, J. A. Woodburn (Members of Council), H. D. Bell, T. F. Considine, J. Henderson, K. Leinberger, E. Neufeld, S. Newton, J. F. Pyles, W. Simpson, R. A. H. Flugge-de Smidt, J. J. R. Smythe, J. Thorlund, and H. R. S. Wilkes.

5 Associates: Messrs. C. E. Deakin, O. A. Gerber, R. J. Kethel, H. Rusden, and W. Russell.

2 Visitors and H. A. G. Jeffreys (Secretary).

MINUTES.

The President announced that, as the *Journal* containing the Minutes of the September Meeting was still in the hands of the printers, due to editorial delays, he was unable to ask members present to confirm them, but hoped to have permission to do so at the next meeting.

NEW MEMBER.

A ballot was taken for the election of a new member, and the following gentleman was declared unanimously elected:—

Wood, A. A., Consolidated Langlaagte Mines, Ltd., P.O. Box 15, Langlaagte: Assayer.
(Transfer from Associate Roll.)

RESEARCH ENDOWMENT FUND.

The President said he had been requested to state by the Research Endowment Committee that certain funds were now available for research, and applications would be received for the small funds at their disposal with the object of furthering research.

PRESENTATION OF MEDALS.

The President remarked that the gold medals to be presented were now ready for presentation. He would ask Mr. White to introduce the recipients.

Mr. H. A. White apologised for the absence of a larger audience to welcome the recipients of the awards. Some little criticism had been floating round the community in the past few weeks as to the medals that were to be distributed that evening. That criticism, in his opinion, of course, was entirely unwarranted. The Committee was absolutely unanimous in connection with every recommendation made. He thought the origin of the criticism was due to the fact that it had been overlooked that, not only were the medals intended as a reward for papers that had been presented, but their real and basic object was the encouragement of research. If a paper showed that the right spirit of research was in the author of it, although he may not have succeeded in expressing himself in such a way as to meet with the approval of the older hands in research work, yet, if he had the meat and the substance in that paper—they were not looking for minor defects, but for the basic promise, and they had granted medals in certain cases perhaps on those lines.

The first gentleman whom they would now ask to come forward and receive a medal was Mr. R. A. Cooper. He needed no introduction from him (the speaker) to those present because, in addition to the papers he had shared in producing during

the year he had given other valuable contributions to the Society. One of his earlier contributions was undoubtedly in flat opposition to some researches of his (the speaker's); but he did not suppose members would consider them of any less value because of that. Therefore, it was with very great pleasure that they could now call upon Mr. Cooper to come forward and receive his medal, because, not only had he earned it that year, but had deserved it in the past.

The President, in handing the medal to Mr. Cooper, offered him his hearty congratulations.

Mr. H. A. White said the next recipient of a medal was Mr. W. T. Heslop. Mr. Heslop was, unfortunately, unable to be present. Members would recollect the very able paper given before the Society by Mr. Heslop on a question concerning coal mining—and about which subject he (the speaker) was supposed not to know very much. Therefore, he need not say anything. He could only say that, in the opinion of the Committee, that paper was thoroughly deserving of a medal. He would ask Mr. John Watson to receive the medal on behalf of Mr. Heslop.

Mr. John Watson, in apologising for the absence of Mr. Heslop, said he could claim to be one of his oldest friends. He would have much pleasure in forwarding the medal on Monday to Mr. Heslop, by registered post. He did not think anyone had worked harder in the interests of scientific coal mining than the recipient of the award.

Mr. H. A. White said Mr. E. P. Reim, living so far away from them, was also unable to be present; but it gave them peculiar gratification in awarding a medal to Mr. Reim, because of the subject with which that gentleman had dealt in connection with South African coal. They had heard it from no less an authority than the Premier that the future of this country was more likely to depend on its future coal supplies than on its gold mines. Mr. Reim had thrown some new light on coal problems; it showed that he was imbued with the proper spirit when he gave the best methods of dealing with all the main varieties. When they in this country were looking forward to the development of industry in order to relieve the depression from which

the country was suffering, they were one and all bound to do all they could as far as lay in their power, and research was a powerful means that must be encouraged.

The next recipient of a medal, said the speaker, was Mr. F. W. Watson, who, of course, needed no introduction from him—in fact, possibly Mr. Watson was an older member of the Society than himself. He was certainly quite as old a researcher. In his particular case there was no need to give a medal to encourage the research which was inborn in him. In this instance, they could look upon it not as a reward but as an acknowledgement of the services he had given not only in connection with the papers this year but in connection with his previous papers given before the Society.

The President, in presenting the medal to Mr. Watson, said he had great pleasure in doing so.

Mr. F. W. Watson, in acknowledging the award, said he must thank the Research Committee and the Council of the Society for the award, and Mr. White for his very kind remarks. Mr. Cooper, on receiving his award, had not said anything; but he (the speaker) would like to express his appreciation for the assistance he had received from Mr. Cooper and Mr. Bell during the past 2½ years. Mr. Cooper's work on osmiridium had been indefatigable, and, he was afraid, in maintaining silence, he had not expressed what he should have done. He had taken considerable risk in connection with the investigation, and they had had to clear out of the laboratory more than once on account of the fumes. Mr. R. D. Bell had rivalled Mr. Cooper in this respect in connection with his search for rosin. Acetic anhydride is not a pleasant companion—but Mr. Bell had separated and identified rosin in spite of the fact that manufacturers had said it was not present. He was very pleased that the members of the staff of the laboratory had been associated with him in those awards.

Mr. H. A. White said one further award had been made by the Council; that was, in the case of Mr. Bell, to whom Mr. Watson had so eloquently referred, for his assistance in writing a very valuable paper in which he was joint author with Mr. Watson. They could not give him a medal, on account of his being a joint author, but the Research Committee felt it was their bounden duty that he should have an award;

they certainly recognised his ability in connection with the paper and his share in it, and had awarded him a Certificate of Merit.

The President, by way of supplementing Mr. White's remarks, said he was quite sure that Mr. Bell was capable of winning a medal on his own. He that time had come up for a Certificate of Merit, but he hoped that on a future occasion he would come up for the medal itself.

GENERAL BUSINESS.

MINE AMBULANCE COMPETITIONS.

The President announced that the Mine Ambulance Competitions were held last Sunday. These competitions were inaugurated by the Society in 1912, when the Society offered for annual competition two shields, one for underground workers and one for surface workers on the metalliferous mines. The shields were competed for by ambulance teams of five men, one team being allowed for each 100 or portion thereof white men employed on any mine. A silver medal was also presented, and a bronze medal to the second team. He was present at the competition and had been very much struck with the extreme keenness shown by the various teams entering the competition. In connection with the surface competition, Modder B had carried off the prize with 85 per cent. of the points. The next were the Knight Central, Wit Deep, New Modder, and Village Deep. In the underground competition the Crown Mines West were the most successful; they had 78.5 per cent., but were very closely followed by the Consolidated Main Reef, who had 77.5 per cent. As showing the keenness with which the competitions were carried out there was one team—he thought it was from Geduld—which was keen enough to hire transport to bring them here so as to be in time for the competition. That, of course, showed that if an accident did take place on those mines there was an organisation for giving first aid and rendering the lot of those who had suffered as painless as possible. The competitions were carried out in a very sportsmanlike way. The doctors were very pleased with the results, and the management of the Robinson Deep were very excellent hosts.

He would now call upon Mr. Watson to give a report on the Vereeniging Excursion.

EXCURSION TO VEREENIGING.

Mr. F. W. Watson said they had had an excursion to Vereeniging on the 7th October. Unfortunately the attendance was small. There were one or two reasons for that. Their President could not be there owing to important events in connection with the University. One of their Vice-Presidents—Mr. H. R. Adam—and several others were unfortunately unavoidably unable to be present owing to the same reason. A feeling prevailed that these excursions on a Saturday morning were unfortunate; that generally the works were closed down by the time members and their friends arrived there, and they did not see the working operations. This had happened twice within his experience—once at the Starch Works and once at the Brick and Tile Works. However, the excursion was a most enjoyable one. Their hosts had shown them all they could and treated them royally. He thought the excursion was instructive and of educational value. In view of the prominence now of the steel industry he thought all those who went down to Vereeniging were well repaid for the trouble they had taken in going that distance. He thought they ought to put on record their very hearty appreciation of the directors of the Vereeniging Estates for their kindness and courtesy to them.

NOTES ON ORGANISATION FOR PROMOTING INDUSTRIAL EFFICIENCY.

By S. DE SMIDT.

(Printed in *Journal*, August, 1922.)

The President asked Mr. de Smidt if he would introduce his paper published in the August *Journal* under the above heading.

Mr. de Smidt, in introducing his paper, said his "Notes on Organisation for Promoting Industrial Efficiency" was before members. There was no need for him to read it. It had been published in the August number of the *Journal*. He would just like to add that in one group of mines there was in existence a Technical Investigation Committee and an Ideas and

Suggestions Committee. The establishment of these Committees was a step in the right direction; but he wanted to make a plea for a freer interchange and discussion of ideas and suggestions and also of the results of investigations. As gold mining was not a competitive industry, why should not ideas, suggestions, and the results of investigations be published in our mining journals and newspapers for the common good? He was sure that for every idea one group of mines gave to the mining industry as a whole they would receive another in return from another group. In that way their advance would be more rapid than when each group was working in a water-tight compartment and jealously guarding their discoveries, as in the case of a competitive factory.

The President asked members present to pass a very hearty vote of thanks to Mr. de Smidt for the time and trouble he had taken in connection with the preparation of his paper. It was one which ought to lead to a very large amount of discussion. From his (the speaker's) point of view, he thought it would tend to a very much larger number of the younger generation going in for mining. At present, as they knew, on account of phthisis on the mines combined with the idea that promotion was not sufficiently rapid, they could not get young men to go in for mining engineering in the same way as they did, say, 10 or 15 years ago; and how the men who were leaving the industry were going to be replaced in 10 or 15 years' time it was very difficult to know, because there did not seem to be the incentive for them from the industry to take up the profession. If, as Mr. de Smidt had suggested, they had a large number of efficiency engineers in each group, there was no doubt about it that the good which would result to the mines themselves would be exceedingly valuable. He would, therefore, ask them to give Mr. de Smidt a very hearty vote of thanks. The subject was now open for discussion.

Mr. C. J. Gray said he would like to join in the appreciation of the paper. Mr. de Smidt had travelled, had observed, and beyond that had given them the benefit of his observations. One might doubt whether the scheme of mining engineers, or efficiency engineers, could be applied directly on the mines with advantage. There was a chain of responsibility on the mines. If they introduced a man working directly under the

general manager and allowed him to interfere in the different branches of the work, alternately concerning himself with the departments of the reduction officer, the underground manager, the mine overseer, and the shift boss, they were expecting an amount of tact and goodwill which perhaps they could hardly expect from ordinary mortals. Unless the scheme could be directed in such a way that each person under whom the efficiency engineer was working would have a direct interest in his work—that he could claim some credit for any discovery or improvement that was made—there would be considerable danger that the advantages gained in one direction might be lost in another. It was not beyond the skill of those persons who would organise the industry to overcome those difficulties, but it was an aspect of the scheme which they should observe very carefully.

The great point was the author's wish for the introduction of a larger number of technically educated men, men with University or School of Mines' education. He wished to give those men time and opportunity to investigate efficiency in different branches of mining. As it was now, most men with the necessary knowledge and training had not the time or opportunity to go into the details of the work thoroughly. They had to do the work of the day. They could not take a wide view, and could not try experiments to the extent it was desirable they should. From that point of view the introduction of these efficiency men or "mining engineers" would be of great advantage. There was, at the present time, a shortage of technically educated men on the mines, while very few, relatively speaking, were being educated for the work which has to be done in the future. The chief reason for the small number of students was the fear of miners' phthisis. There was the direct fear that a man might contract it and lose his profession, but there was also the difficulty that he had to get a certificate from the Miners' Phthisis Bureau before he could work underground. A student might spend years on study and then be denied that certificate. He understood that in one year six students from the second-year mining course at the School of Mines went up to the Bureau, and, of those six, five were advised not to proceed, as at the end of their course they would probably be unable to get a certificate to work underground. It was a very serious matter. If they could only get control over miners' phthisis so

thoroughly that it could be recognised that the risk in the mines was exceedingly small, then it would no longer be necessary for the Bureau to require that the young men who go to work in the mines should be of such fine physique as they do at present.

While they had the difficulties of miners' phthisis and of educated men not being satisfied with the positions that are given them in the mines, they would have a shortage of competent candidates for those positions. To get over that shortage, at present—before they could get rid of phthisis—the only thing that could be done was to make the inducements for those men greater; that was, to arrange positions suitable for them, and to give them pay perhaps a little higher at first than they actually merited. There was the case of the young man who had come from a mining school, who thought his knowledge was exceedingly valuable and that he should receive considerable pay. He found when he came to a mine that he had to draw lower pay than a man who had simply worked in the mines for two or three years; well, probably the fact was that the School of Mines' man was not worth more in the mechanical work he had to do, or not worth even as much, as the man who had not had his education but who had been doing the work practically for a short while. Of course, the advantage that the School of Mines' man had was that he should advance more rapidly; he should go up faster and be able to rise further than the man who had not had the same technical education: but that was something which the young man hardly realised; he wanted to be recognised in the first place. So, if something could be done to improve his starting position they would get a larger number of men endeavouring to obtain mining education. In the future they would require a larger number of technically educated men; but even at the present time he thought there were particular directions in which full use was not being made of those available. They had now the problems of ventilation, of which he would be speaking later, in connection with miners' phthisis. The ordinary practical man in the mine, who had simply worked up from the miner, had a very hazy idea of ventilation. He could not adapt himself to control most advantageously the distribution of ventilation or to realise fully its advantages. The technically trained man could do that far more easily. Then, again, they were constantly hearing of discoveries of gold in old

mines or in portions of a mine which were supposed to be worked out. Now, that was probably due to the fact that the men who had been working those mines, from the mine overseers downwards, including the samplers, were men who were without higher education; they had very little vision. When the reef was all plain they would follow it and sample it; but, if there was a fault or a dyke, they might go astray, and they did not realise it. The technically trained man with a wider vision of the position would be far less likely to leave reef unseen. Such losses of reef were, though unrecognised, doubtless very serious to many gold mining companies. Of course, everybody present knew there were further fields of work in which technically trained men could be employed with advantage, but he need not mention those now.

As far as the members of the Society were concerned, they had a great deal to do with the work in and about the mines; and, if they would realise the advantage which could be obtained by using those younger men and giving them an opportunity to rise, they would have a larger supply, and the mines would gain in the long run.

Mr. A. Whitby said there was one point that Mr. Gray had not mentioned; that was, the position of the assayer on the mines. It was a very important one, and yet the class of man doing the underground sampling could hardly be called a technical man, at all events the assistant samplers. He also wished to point out another little matter, where the connection between the surface work and the underground work was also of importance—that was, in the distribution of the clean-up of stopes and drives. The underground manager or the manager who had control of the whole thing usually took those clean-ups at the most irregular intervals, and never told the reduction works officer anything at all about it. That was most serious and highly technical, because they had all sorts of organic matter, acids, and other material to deal with which have an adverse effect on cyanide treatment.

Mr. H. Pirow, in associating himself with the vote of thanks to Mr. de Smidt, more particularly from the point of view mentioned by the President with reference to arousing discussion in connection with efficiency on the mines, said he, personally, had read Mr. de Smidt's paper in that way;

that rather than putting forward a scheme for introducing efficiency engineers into the mines Mr. de Smidt was bringing up the subject with the object of arousing discussion and having the pros and cons thoroughly thrashed out. He might mention one or two matters which had struck him as likely points for discussion.

He thought one realised that efficiency was dependant on two main considerations: the first one was the equipment provided, and the second the efficiency of the worker himself. As regards the equipment he thought they could say that they on the Rand saw signs of very satisfactory progress. One need only consider, for instance, the latest drilling machines, pumping stations, and so forth, to realise that. There were, perhaps, one or two branches of mining where this progress was not quite so evident. He might be permitted to mention one; that was, in connection with the matter of cleaning out stopes, and development; the more so as it appeared that in other countries different methods had been adopted; and that gave rise to the question whether a modification of our methods might not be in place.

Then, there was the other point, that of the efficiency of the worker himself. It struck him that there were two main directions in which they could strive after increased efficiency: the first being the training of the worker, and the second the conditions under which he worked. They were very fortunate here in possessing a School of Mines, on the one hand, and the Miners' Training School on the other. He thought Mr. Gray had dealt very fully with the first. There was, however, another consideration which arose; that was, whether one should not strive after somewhat better training for the native as well as the white man. Of course, against that it would be argued that the time service of the native on the mines was very limited; but, on the other hand, it seemed reasonable to expect that, if some regular scheme of training could be provided, the native would tend to become a more permanent worker on the mines than he was at present.

With regard to the second point which he had mentioned, namely, the health conditions under which the worker had to perform his duties, a lot of research work had recently been done on that, and they were indebted to Dr. Orenstein and Mr. Ireland

for a paper on that subject. It had struck him that there, too, was a field in which further research was possible. It had been accepted as an axiom that a happy and healthy man was always a more efficient worker, and he thought that any capital expended in that direction would show handsome dividends.

Mr. J. Chilton said he too would like to associate himself with the vote of thanks to Mr. de Smidt. If he recollected rightly the writer in his paper did not give a definition of efficiency. He (the speaker) supposed there were hundreds of definitions, but the one that pleased him most was this: "Efficiency is the elimination of waste in men and materials." In an engineering works such as Mr. de Smidt describes in his paper there are ideal conditions for efficiency, as every worker is directly under the eye of the foreman, but when an attempt is made to apply efficiency methods to underground work there is quite a different problem to be faced. Mining men work in isolated spots in tunnels, drives, winzes, and stopes, where close observation is impossible, consequently supervision must be slack when compared with conditions obtained on the surface, and the most competent shift-boss can only exercise an intermittent control.

In his opinion before the efficiency engineer, which Mr. de Smidt so eloquently advocates, there should be the trained observer to study mining methods and the waste in time, materials, and men. Some steps in this direction are being taken on some of the larger mines, and if report speaks truly the waste of time shown in some instances is surprising. Until a time study has been made of every mining operation under every conceivable condition, the efficiency engineer would be wasting time. Before he comes there must be a lot of spade work done, and this could be accomplished by intelligent observers leaving the matters of co-ordination to the more highly paid efficiency engineer.

Mr. S. Newton considered that a certain amount of natural curiosity should accompany efficiency. In the near neighbourhood of the Modder B they had a mining school. He had approached the principal and told him how delighted he would be if he would bring half-a-dozen of his boys along one afternoon to have the process of extraction explained. The interest

in their work underground would be enhanced by the knowledge of what was going to happen to the rock after it had been broken. These young fellows were in the early stages; some of them, no doubt, might fill the bill as mining engineers in the future; but the combination whereby, after they had been underground, they were shown round the surface one afternoon a week, would certainly give them the ground work to enable them to take more interest in their work.

Mr. J. Hayward Johnson remembered a year or two back when they had a native strike. The Aurora West, at that time, was a training school for miners. Quite a number of these young fellows were brought down to the reduction plant and started doing the work of the natives. It had surprised him very much to find the interest that these men took in gold recovery. He had asked several of them why they did not take more interest in this branch of mining. They replied, as a matter of fact they would like to, but that by the time they had done their shifts and had come up from underground they felt too worn-out to take further interest in surface work.

One thing that rather spoiled the efficiency on these mines was the position of the native labourer. He came in on contract. As long as he went underground and did a certain amount of work, well and good, but if he did not do his work he could be punished in the Court. That was a line of action people did not like to take because it meant a waste of time in attending Court. If they had white, unskilled labourers there would always be the fear of being discharged for inefficiency, but the native labourer has no fear of being discharged; his services are in demand and his independence is great; he does not understand that he must do his best, for he has either no ambition or no incentive.

The President hoped the paper would provoke much discussion in the future; it seemed to have aroused a fair amount of interest.

He would now call on Dr. Lehfeldt to introduce the paper which he had been good enough to write for them, and which had been already published in the *Journal* for August last.

STATISTICS OF EXTREMELY DEPRECIATED CURRENCIES.

By R. A. LEHFELDT.

(Printed in *Journal*, August, 1922.)

Dr. R. A. Lehfeldt said the paper, which had been published in the *Journal*, appeared to be very technical. He did not think it was desirable to take up space in the *Journal* by explaining technical terms, which made the paper look somewhat obscure. He was quite aware, however, that he was not addressing an audience of economists. Therefore, he hoped they would not think it a waste of time if he now explained the technicalities involved. They would see it was a matter of common-sense. The paper starts out with some remarks about the elasticity of demand for money. That was a technical term that wanted explaining. The elasticity of demand was an expression as to how much the demand for anything was affected by change in price. For instance, supposing the price of tram fares was reduced from 1.1/3rd pence to 1d., the question which interested the Tramway and Lighting Committee was, how many more people were going to ride in the trams? That relation could be definitely expressed by that quantity which was described as the elasticity of demand. It is the fraction in which the denominator is the change in price expressed as a fraction; the numerator was the corresponding fractional change in quantity—the quantity of tram rides, in this case. Thus, supposing that the price of tram fare was reduced 10 per cent.; if that made people buy 10 per cent. more of them the receipts of the Tramway Department would be the same as before. If it made them buy 20 per cent. more, the receipts would go up. If there were only a 5 per cent. increase in traffic the receipts would go down. In this instance the change in price was what one thinks of first.

Take another case. Supposing there were a good harvest, the price of corn went down; if there were a bad harvest the price went up. Therefore, it was the numerator which acted first. If the harvest this year were 10 per cent. larger than last year, the question was, how the price would be affected? They would see that that was a quantity of very considerable importance, and that

was what was described in books as the elasticity of demand. He would just make a slight alteration on the blackboard; he would cross out "price" and write "value," because "price" was rather apt to be confusing when they were talking about money.

Now, the usual doctrine about money was that if the quantity of it were increased by a certain percentage and no other changes took place, the value would be reduced by the same percentage, which would make the fraction come out to "1." So it might be put technically that the elasticity for the demand for money was 1. Supposing they had ten millions worth of money in this country and had a certain amount of trade to carry on; the money was simply to facilitate transactions—to pay bills. Supposing nothing else having changed, they had 11 millions worth of money in the country; what were they to do with it? They had the same amount of corn and iron and boots and doctors' bills, and so on, as before; the only thing possible to do with it really was to pay more for everything. That was what happened. If you have a certain amount of copper in the country and that were increased by 10 per cent. you could at least make some more copper pots with it; but if the amount of money is increased the total value of the money must be the same as before. This is the accepted doctrine, and is true enough with regard to ordinary changes in the amount of money by a few per cent.

They had recently witnessed changes that were not normal—and not a few per cent.—in the amount of money. When they got ten times as much money in a country, or 100 times as much money, it did not necessarily follow that the value dropped in the same ratio. Prof. Cannan had recently written a paper giving reasons why the value should drop more than in proportion. His own paper was suggested to him by reading Cannan's paper; it had made him think about it. He (Prof. Cannan) had merely put the matter in a general way; but it seemed to be a matter which could be discussed statistically. They could make an attempt to work out the fraction on the board and see whether it was less than unity. As Prof. Cannan had said, there were certain important consequences to be derived from that, but he was not going to prove them there that night: it was shown in the paper. But one consequence had not been pointed out.

Supposing they tried to estimate how much value Governments got out of the public by offering additional paper money to them; supposing they started in South Africa when the bank note was at par—it was not, of course, just now; but supposing it were and the Government were to print a million more bank notes and hand them out in payment of bills—he expected the public would take them at par—well, that would be one million pounds. Supposing they printed another million; very likely the public would begin to get tired of it and the bank notes would drop in value. Supposing they issued a million-and-a-half more; it would drop a little more, and so on. If the statement that the elasticity of demand is equal to 1 held, then it could be proved mathematically that the value which the Government could get out of the public in that way was infinite. If the elasticity possessed any particular value—as a matter of fact, it worked out to $e/(1-e)$ times the original value of the currency; he would not bore them with the details; if e was anything appreciably less than 1, it worked out to three or four or five, or something like that. It might lead to a value three or four times the original value of the currency. Now, how much currency was there in the country to start with? Well, in Germany before the war, there were something like 200 or 300 millions sterling in currency. If that estimate was right—assuming there was a reasonable value for " e ," the amount of value which the Government by that device could practically borrow from its subjects was limited to a certain number of times that original value of the currency. What had happened in England was, the Government had borrowed about 7,000 or 8,000 millions by the ordinary methods of placing loans on the market. They had not adopted that plan of inflating currency to any appreciable extent. It was an interesting point to know how much could be got out of the public in the way he had mentioned.

Another interesting point was to see how the value of money was affected by increasing the amount of it. He had looked up the monthly bulletin of the League of Nations to see the amount of paper money. Let them take Austria, for instance—there was nothing about Russia; it would be interesting to get the figures—the third column gave them the amount of notes in circulation expressed in 10⁹ crowns; the ordinary

ways of expressing quantities of money were hardly sufficient. Then, side by side, was given the value of Viennese exchange expressed in the percentage of gold, as the American exchange was still at par with gold. For instance, in June, 1920, the Austrian exchange was 3.632%; that meant that 100 crowns, Austrian paper money, were worth 3.632 golden crowns. Now, if they took those two columns, the one showing the number of crowns in circulation, and the value of the crown, and multiplied them together, they got the total value of the currency expressed in terms of gold. Of course, it was quite true that the value of gold had not kept constant; but he thought they need not trouble about that.

Now, if the statement were true—if the value of the unit only fell in proportion to the increase in numbers, then the product of the quantity into the value would remain constant. It did not remain constant. The best way in which to see that was to look at the diagram (exhibiting). The only figures he could get for the four years were those of Germany. The Polish and Austrian figures started there (indicating). Now, taking the German figures, the value of the currency before the war was about that level (indicating). Anybody who read that paper in June, 1919, would see that the exchange value of the German currency then was about twice as much as during the war. One might easily have concluded it was going to drop. At that time, a few months after the armistice, the value of German paper money far exceeded what was at all reasonable as regards the quantity of it. Things began to settle down towards the end of 1919.

He wanted to draw attention to two features of the diagram; firstly, as regards the oscillations—there is a period of about 10 months. That was to say there was a slump in the depreciated currency once in about 10 months. Of course, it might be purely accidental, or the result of historical incidents, or changes of sentiment, sometimes pessimistic, sometimes optimistic.

The other feature was the gradual trend downwards—that was to say, the whole value of the currency of the country had obviously fallen off—and that meant—reverting to the fraction on the board—that the change in value exceeded the change in quantity. So that, if they took the paper money and doubled it in amount, they not merely halved it in value—the individual

notes—but they reduced them $2\frac{1}{2}$ or perhaps three times; with the result that the product of the number of units into the value of the units became less than before. That had happened in Germany.

Germany before the war had an amount of gold and paper money in circulation worth about 250 millions sterling. They could buy the whole of the currency of Germany at the present time for less than 100 millions. That was an enormous drop in value, and appeared to be associated with that extreme increase in the number of nominal units—that was to say, it had gone down beyond the range at which the increase in quantity was compensated by a reduction in value. The public was evidently getting so suspicious of the currency that it dropped more than they could expect from the increase in quantity. Austria and Poland had gone further in that matter. Poland was making some attempt to restore its finances. It did not show it very much on the diagram. If they looked at the diagram they would see the two lines for Austria and Poland were very much alike; they were expressed in the same unit. The line for Germany was not the same; the unit was 10 times larger. If they took those for Austria and Poland they would see the two lines ran closely parallel, both as to the oscillations and as to the general trend downwards; and that, although Poland was on the side of the Allies and received a certain amount of political support, it did not make very much difference. It was rather in the nature of a natural phenomenon. That was really about all he had to say in connection with what he had put into his paper, except that from the general course of the figure they could calculate the quantity "e" elasticity when they would find that it worked out to about .73 in Austria and .67 in Poland, say .7; but in the case of Germany it worked out at about .5. His figures only went up to May last, when German currency was still over 1 per cent. in value. So that there appeared to have been an influence exerting itself in the case of Germany which did not show in the case of Austria and Poland. If they took the value of .7—(explaining on blackboard)—it showed that the paper money would yield only about two or three times the value of the currency previously in circulation; whereas those countries which had borrowed in a straightforward fashion in the open market had borrowed larger sums than

that without dislocating their currency. So that was an additional condemnation of that very undesirable method of raising money.

The President asked members present to pass a very hearty vote of thanks to Dr. Lehfeldt for coming there that evening and giving them that extremely interesting explanation of the valuable paper which he had written for their *Journal*. Economics was not often discussed in the Society; but, at any rate, the explanation had taught them a great deal. An interesting point was the periodic rise and fall in the total value of the currencies. If there were any questions they would like to ask, Dr. Lehfeldt would be very pleased to answer them.

Mr. Rusden supposed there was no connection between the gold premium and the rise and fall of the German mark.

Dr. Lehfeldt replied that he did not think so; and that it was exclusively from the value of English currency.

Mr. John Watson referred to the slump in escudos in Portuguese East Africa. He would like to ask if it were a parallel case to that with which Dr. Lehfeldt had dealt that night.

Dr. Lehfeldt said he had not examined the statistics about Portuguese currency. It had not depreciated to the same extent as had the others. At present there were over 100 escudos to the £, whilst the par rate was $4\frac{1}{2}$ to the £. The range of values in the case of Germany was from about 10 or 15 per cent. down to 1 per cent., and in the case of Poland and Austria it started at about 3 per cent. and went down to a fraction. So that the Portuguese currency had not quite descended to those levels. It might, of course. He dared say if they worked out the statistics they would get some results on the same general lines.

Mr. H. R. Adam gathered from Dr. Lehfeldt's explanation that the Governments of these countries got a good deal out of their subjects by the device he had mentioned. He wanted to know whether the people were taxed in addition. Did they get it out of their subjects in two ways—by way of income tax and poll tax in addition to the device? Or how were the subjects of those countries off as compared with a very heavily taxed country, such as England was

at present? Which device was easier or harder on the subjects of those countries?

Dr. Lehfeldt said, of course, the attitude of those countries of which he had been speaking was that they could not get enough out of their subjects by way of taxation. They all professed that they taxed as heavily as they could venture to do, but still they had not got enough to pay their expenses. He could not tell them what the facts were; he could not say just how taxation in Germany or Austria compared with the taxation in England; it was a disputed point, and it was rather difficult to get at the facts.

The President asked Dr. Lehfeldt whether it was possible to fit theoretical curves to the actual curves of experience?

Dr. Lehfeldt replied that he did not think it was worth while trying to adopt any elaborate methods. They had to represent the oscillation as well as the general trend of the thing, which, he thought, would be giving a nominal accuracy which did not correspond with the facts. They would see, there were only three cases which he had been able to investigate over two or three years, and to enlarge that into a generalisation would be going beyond what the facts justified. The general downward tendency thereof was in accordance with the view that the elasticity had a certain value less than unity, and, if they assumed it was a constant value, like .7, for instance, and put that into the equation, then they did get a definite curve. Let them take the equation on page 1, second column, the third equation down; it looked very much like an equation in Thermo-dynamics—an equation between P and Q, where P was the value of the currency and Q was the amount thereof; the only arbitrary constant was e. If they took the value of e they could work out the equation, which would be mathematically similar to the equation they got in books of Expansion Tables. But that would not be the curve drawn in the diagram because that curve was between value and time. But they could take the observations, plot them on a curve, showing the relation between P and Q, and compare that with a curve calculated from the equation.

Mr. H. Pirow asked whether Dr. Lehfeldt could tell them whether there was any direct relation between the price paid in

paper marks for an article in such a country with the rate of exchange. He had come across friends who had bought articles in Germany, say, for instance, an overcoat, for a certain sum, and a few days later nearly three times the price was asked for the same article. Was that merely due to panic, or were there other factors governing it?

Dr. Lehfeldt, in reply, said that raised rather an extensive question which was not included in the subject of the paper. The internal purchasing power of money in countries like Germany did not follow closely the external. The fall in the value of the mark—the exchange value of the mark: that was to say, the amount of gold they could get for it, or the amount of sterling they could buy in the foreign exchange market fluctuated; it was very, very sensitive to all sorts of influences; it was a very rapid and sensitive market; whereas the price which was paid for things in the shops in Germany did not change in amount nearly so rapidly. Of course there might be cases like the one suggested by the questioner, but he thought it would be rather a panic case to rise three times in price in three days. But the general state of things was that the price inside the country lagged very much behind the foreign exchange value. The effect of that was when the foreign exchange had fallen the price of articles in the country did not rise proportionately; therefore the gold value of those articles was less. They would find, of course, innumerable examples—he had heard of one in the tram that night of a gentleman who had recently bought a pair of boots in Lourenco Marques for 15s.; that was because the exchange value of escudos had dropped very rapidly of late; the price of boots in escudos had not risen proportionately. People were always picking up bargains in depreciated currencies for the reason that there was a very decided lag in the internal price. But it was a temporary phenomenon. If the foreign exchange were to become stabilised eventually the internal price would become stabilised in proportion.

A Visitor asked whether a point could be reached when the lack of confidence in the currency would divorce the quantity from the value altogether, in such a way that the increase could practically have no effect? For instance, with regard to the statements in the papers lately that the gold treasury mark notes had been issued for the use of

industrialists in Germany, would not that cause such a loss of confidence in the mark in the ordinary market that the relation between the quantity and the value would be separated?

Dr. Lehfeldt took it that the end of it would be even a more rapid depreciation in value as compared with quantity. The issue of paper marks which are exchangeable for gold would naturally tend to make still worse the situation of those which were not exchangeable for gold. The whole tendency was for the value to drop more than the quantity increased, and that was expressed by the value of *e* being less than unity. But, if that got exaggerated and the value of '*e*' dropped to still lower values than .7 or .5, it simply meant the mark would drop in value more rapidly. He would say that situation had occurred in Russia; people would begin to regard paper money as having no value. One wondered how much longer they would go on accepting it. He would not recommend anybody to buy kronas or marks for long periods.

SYMPOSIUM: MINERS' PHTHISIS.

Mr. C. J. Gray said he had been asked to read a report by a Committee which had been dealing with certain matters in connection with miners' phthisis. He would first explain how the Committee arose.

Inspectors of Mines found the conditions in development workings in the mines unsatisfactory in regard to dust. After blasting when the night shift went into the drives, although the water-blasts might have worked and everything seemed to be in order, there was still almost always a very considerable quantity of fine dust suspended in the air of those drives. It was very fine, and gave high counts with the konimeter, though it would not weigh much in the sugar-tube test. Conditions were similar in the drives on the day shift during the drilling, though apparently the use of water was all that could be desired.

A Committee of Inspectors was appointed to investigate those conditions, and, as the result of a very considerable number of observations, a report was drawn up suggesting that drastic measures were necessary to improve the conditions.

A Conference was then held between representatives of the Department of Mines

and of the Chamber of Mines, in which a question arose as to whether some of the conclusions should not be checked by special tests. It was then arranged between the Government and the Chamber of Mines that a Joint Committee should be appointed to carry out those tests. The Committee was appointed a little over a year ago; Major Swinburne, Chief Inspector of Mines, was Chairman, and there were three other Inspectors of Mines on it, viz., Mr. Pretorius, Mr. Ray, and the speaker. As representing the Chamber of Mines, there were Mr. Coaton, Manager of the Springs Mines; Mr. Payne, Manager of the Consolidated Langlaagte; Mr. Lange, Sectional Manager of the Crown Mines; and Mr. Boyd, of the Dust Sampling Department of the Chamber of Mines.

The Committee had completed its work, and the following was its report. It was dated the 16th September, 1922, and was directed to the Government Mining Engineer and the President of the Transvaal Chamber of Mines:—

“On the 18th August, 1921, we submitted an interim report containing the following four conclusions:—

- “1. That observance of the water-blast regulations will not necessarily give satisfactory conditions in development ends.
2. (a) That the useful purpose of a water-blast is to damp the face and broken rock.
- (b) That the present water-blast is ineffective in allaying the fine dust which has been projected by the blast.
- (c) That, if the water used is not clear of solid particles, the water-blast is likely to increase the dust contents of development ends.
- (d) That one water-blast at the face is sufficient to damp the face and broken rock if it is of a type to project the necessary water.
3. That the exact amount of air required to clear a development end has not been definitely determined. From tests conducted it would appear to be not less than a volume of air equal to that of the development end. The exact amount may be dependent upon the time interval and upon the rate and method of supply of the air.

4. That the most practicable way of dealing with the fine dust in development ends is by ventilation.”

Subsequent to submission of that interim report it was found that, under certain circumstances, fine particles of soluble salts derived from the mine water supply accompany other particles upon konimeter slides and are counted as silica. It was therefore considered advisable to repeat a great deal of the work upon which the report was based, using in that check work, as in our further investigations, a process for temporarily eliminating soluble salts by treating the slides with acid vapour.

The check work has confirmed all the conclusions given in our interim report, although it has shown that the waterblast is not quite so ineffective as indicated by the first tests, and that in exceptional circumstances convection air currents may aid special ventilation in clearing dust from development ends.

Further tests have been carried out to endeavour to ascertain more accurately the amount of air necessary for ventilation of development ends after the blast and to ascertain the amount required during the working shift.

As the result of its work, your Committee is of opinion:—

re Dust from Blasting.

- A. That the amount of air used for ventilating a development end before entry after blasting should be not less than the volume of that end beyond the last through ventilating connection; that, to meet special conditions in which the full volume would not be necessary, the Inspector of Mines should have authority to reduce this requirement.
- B. That the present regulations re water-blasts should be so modified as to require only one water-blast in a development end; such water-blast should discharge within a distance of not more than fifty feet of the face which is being advanced and effectively damp the face and broken rock.

re Dust from Drilling, etc.

- C. That, in order to obtain satisfactory conditions during the drilling shift in a development end, it is necessary to supply considerable quantities of air

for ventilation, in addition to the air supplied to the drill.

D. That the necessary quantity varies so greatly that it is impracticable to lay down definitely the amount of additional air required.

E. That mines should so improve the dust conditions in development ends as to reduce the amount of phthisis-producing dust found in them to that found in stopes.

The tests have been carried out at the Crown Mines on the Central Rand and at the Springs Mines on the Far East Rand.

Records of all the tests have been supplied to you.

Signed by U. P. Swinburne (Chairman), Chas. J. Gray, L. G. Ray, H. G. Pretorius (Mines Department Representatives), A. A. Coaton, Albert E. Payne, A. V. Lange, J. Boyd (Chamber of Mines Representatives).''

It was thought best to keep the report brief and not to burden it with a large amount of detail, but he hoped they would later have a paper dealing with the experimental work carried out by the Committee, so that it would be available for discussion.

He thought the main principle underlying the report was recognition that they were dealing with exceedingly fine dust; it was dust very little more than a micron in diameter; it would remain suspended in the air for hours, even if the air were still. In practice there was always a certain amount of motion in the mine air, and, therefore, they might say that it would not separate at all. The idea had arisen that the water-blast in working in that dust-laden air would wash out that dust; that the particles of water would catch it and bring it down, and so make the air clean. Experience and experiment showed that was not the case. Water-blasts could work in that dust-laden air and make comparatively little difference; they could get rid of all the relatively coarse dust, but not of the exceedingly fine dust which was believed to cause phthisis.

There was a recommendation that only one water-blast should be required. Under the idea that they would take the dust out of the air a series of water-blasts was sometimes put in. The regulations required that more than one water-blast should be put in, if the drive, for instance, was over three hundred feet long beyond a ventilating connection. In practice water-blasts, though working well when tried individually during

the shift, were liable to fail at blasting time; each used compressed air and water, and when all were in action a fall of pressure occurred especially at the water-blast at the end of a pipe line serving several. The end water-blast therefore often failed to wet the face and broken rock, and dust was raised when men who went into the place to clean it out started watering down. The idea in the report was that the water-blast should effectively wet the face and broken rock so that dust should not be raised when the new shift came on, but not that it should deal with dust which had been thrown into the air. To deal with that dust the Committee considered ventilation the proper means. Fresh air should be brought in to displace the dust-laden air and to drive it out so that it would be carried away in the general air current of the mine. An interval after blasting was necessary during which there should be no persons on the return side of the working places so that the dust might be carried away harmlessly.

There was a reference in the report to the question of soluble salts. It was found in one case where an old water-pipe was used for supplying the air for the ventilation of the drive that, apart from any dust in the drive from blasting or drilling, there were considerable counts of particles on the konimeter slides. It was plain they were not silica particles; they had a different appearance; but it was not at all clear that if silica were present in addition, a distinction could be made between individual particles. The Committee's attention was also drawn to the fact that similar particles were in some cases observed in connection with drills in which compressed air and water escaped together so as to produce exceedingly fine spray or fog. It was found that those particles came, in the case of the old water-pipe, from a coating of gypsum which had been deposited in the pipe, and in the case of the drill from calcium sulphate in solution in the mine water supply. If such water were thrown into the air in the form of fog and evaporated, the calcium sulphate contained in it would become suspended in the air and be caught by the konimeter like dust, or, if, before evaporation fine particles of water from the fog were caught, they would as they dried deposit their calcium sulphate on the konimeter slide. Sodium sulphate and sodium chloride also occur in some mine water supplies and act like calcium sulphate.

In order to get over the difficulty, Dr. Mavrogordato, Dr. Moir, and Mr. Ray worked out a method in which the konimeter slides were coated over with glycerine jelly, or with prepared fish-glue instead of the vaseline previously used. When examining the slides after use, a small cell containing a scrap of filter paper moistened with hydrochloric acid solution was put over the slide which was then put under the microscope. The hydrochloric acid and water vapour got into the coating of glycerine jelly, or fish glue, and dissolved the particles of soluble salts, leaving only the silica particles with other insoluble particles such as silicates, if present. In their subsequent tests the Committee treated all the slides in that way so as to eliminate soluble salts, but found that it did not alter the conclusions come to in their Interim Report, which was based on observations where that had not been done.

There was reference in the report to the question as to the amount of air which had to be supplied in a development end while a drill was working in that end during the ordinary working shift. Obviously the amount of dust that was formed depended very largely on the kind of drill in use, on the state of repair of the drill, and on other conditions.

The amount of air that was necessary to dilute the dust so as to bring it down to a reasonable proportion, therefore, varied considerably; and the Committee had not come to any conclusion as to the exact amount which should be supplied. It must vary in accordance with the conditions, but it had been shown it was a considerable amount. Let them take, for instance, the Leyner drill, which was so largely used; the amount of fresh air that was required was, in the speaker's opinion, somewhere in the region of the amount of air that was required to run the drill, so it meant a very considerable additional expense if the dust was to be dealt with by compressed air only. It was far cheaper and more efficient if the air could be supplied through large pipes, with fans, as was now commonly done on the East Rand. In the older mines on the Central Rand, where they had smaller drives, there was difficulty in putting in those large pipes and protecting them against blasting and traffic.

Questions had arisen, which were not dealt with by the Committee, with regard to the way in which ventilation should be supplied. If they had fan ventilation with a

pipe, they could either exhaust air from the development end, or blow it in through the pipe. If they exhausted the air, they had the advantage that persons travelling along the drive would do so in fresh air. If they blew the air into the drive it meant that persons had to travel in the dust-laden air coming out. That showed there was a distinct advantage in the exhaust system, but there was another side to the question. If they were exhausting the air the fan would not draw the air from any considerable distance beyond the pipe end. Therefore, there was an unventilated space between the end of the pipe and the face, which had to be dealt with by auxiliary ventilation by compressed air or otherwise, if the dust was to be cleared away from it. If the air were blown in by means of the pipe it would travel a considerable distance beyond the end and clear the face in one operation. It would be seen that the problem was one which had to be considered in each case.

The question arose as to what standard the Committee should aim at; as to the amount of dust which could be left in the air with safety in development ends. That was a very difficult question to answer. The doctors could not tell them exactly how much fine dust in a given volume of air was innocuous. There were also difficulties in determining the amount of the very fine dust present in any place. By more accurate methods in the use of the microscope, by better objectives, by better condensers, etc., they could go on increasing the konimeter counts of that fine dust very considerably. One saw, with a given objective and given illumination, only dust particles down to a certain size. With improved illumination and objective one could see dust that was finer, and could increase the counts—that was, if the fine dust was there. The fine dust was generally there if they were dealing with conditions after blasting, or with some of those drills which produced fine dust.

The conclusion the Committee came to was that the basis which should be adopted was the condition in the stopes. Under ordinary conditions either konimeter or sugar tube samples taken in stopes give lower results than in drives and other development. The Committee considered that the conditions in development ends should be improved so that the development ends would be no worse than the ordinary stopes in the mine. If that were done it would be tackling the portions of the mine in which the conditions

were worst—and it could be assumed that very considerable progress in dealing with miners' phthisis would be made. In all probability, miners' phthisis was contracted in those places where there was most fine dust. If one eliminated those places, then, he could hope that the disease would be conquered. If they were eliminated and men still got miners' phthisis it would be necessary to go farther, but he did not think it would be necessary. That was the position. The Joint Committee had made a great step forward, in recognition that the mines had that dust in the development ends, and that it must be cleared out by ventilation.

THE DEVELOPMENT OF THE BLAST FURNACE.

By J. HENDERSON.

(Printed in Journal, April, 1922.)

REPLY TO DISCUSSION.

I did not expect that there would be much discussion on this lecture. The object of the lecture was to stimulate interest, as efforts are being made to establish seriously the iron and steel industry in this country. Prof. Stanley in his remarks quoted statistics which showed that there was a sufficiently large import of iron and steel goods to warrant the establishment of the industry. Mention has been made by the Prime Minister quite recently of the immense importance of coal mining in connection with establishment of industries. As it takes practically five tons of coal to produce one ton of finished steel from the raw material it will be readily realised what an effect the establishment of the iron and steel industry would have on coal mining. Also once the industry was established outlets for its products would soon be found. For instance anyone who has travelled on the roads of South Africa is painfully aware of the lack of bridges when crossing dongas. As timber for such a purpose is not only scarce but unsuitable owing to the white ant pest, here is one outlet for structural steel which would improve the internal communications of the country. Fencing and pipes for water services would also provide another outlet

for consumption, and in many other directions new outlets would arise, and there is no doubt that once the iron and steel industry is properly established it will be a success and a great asset in the development of South Africa on industrial lines.

FEATURES OF METALLURGY OF A REFRACTORY GOLD ORE.

By F. WARTENWEILER.

(Printed in Journal, February, 1922.)

REPLY TO DISCUSSION.

There has been but little discussion on this paper, perhaps due to the apparent lack of controversial matter disclosed or paradoxically to the departure from usual practice in many of the features of ore treatment described.

Recent reports about the mine disclose the proposed introduction of the flotation process, presumably to simplify the ore treatment. This process, when experimented with some years ago, yielded a bulky concentrate containing the objectionable graphite. Lack of roasting and concentrate treatment facilities to deal with the larger tonnage of concentrate from this source precluded further progress then. Perhaps the ore is cleaner and floatations methods have improved.

Mr. J. H. Johnson's reference to Machavie brings to mind graphite or carbon occurrences in gold ore in other parts of South Africa. At an operating mine near Klerksdorp a considerable streak of carbon is found plastered on the banket reef. It is called carbon, and has no precipitating properties with regard to gold dissolved in cyanide solutions. A similar carbon is found not infrequently in the Witwatersrand banket ore as an infiltrated mineral, and is often associated with high gold content. To the best of the writer's knowledge it does not act as a precipitant of gold from cyanide solutions, judging by the inability to dissolve gold from material collected in the treatment tanks of the reduction works, by means of sodium sulphide. Its proportion by weight is infinitely less than that found in Prestea ore, and its appearance is not that of a carbonaceous shale as at Prestea but more that of a carbon deposit, having the appearance of lamp black.

NOTES ON THE MANIPULATION OF OSMIRIDIUM CONCENTRATE.

By R. A. COOPER.

(Published in Journal, February, 1922.)

FINAL DISCUSSION.

Mr. F. W. Watson said he would like to make a few remarks on this subject with a view to show how complex was the material to be dealt with. One ordinary concentrate, from a mine, was examined recently, and the following were found present: arsenic, antimony, mercury, cobalt, nickel, iron, alumina, vanadium, and phosphoric acid. These were in addition to gold, silver, and platinum group metals.

He considered that it was extremely interesting, and perhaps unknown to many, that all these elements were associated in some instances with the banket and appeared in these concentrates.

The refining of this complex substance should only be undertaken by skilled and experienced workers, as otherwise serious accidents may occur.

The meeting then terminated.

Notices and Abstracts of Articles and Papers.

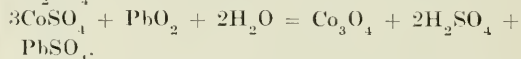
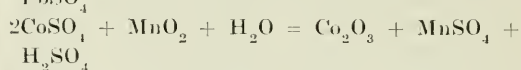
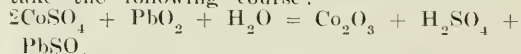
CHEMISTRY

THE REMOVAL OF IMPURITIES FROM ZINC SULPHATE LIQUORS.—*De-ironisation.*—After extraction and separation of the zinc liquors from the residues, iron was removed by raising the temperature of the solution, adding whiting and blowing in air. The operation is termed "*de-ironisation.*" Iron is precipitated as ferric hydrate, the precipitate carrying some manganese and also basic zinc compounds. This "iron precipitate" was used again in reducing the acidity in the extractors to effect the recovery of its zinc, very little or none of the iron passing back into the liquors in the last stages of neutralisation. De-ironisation was facilitated by the presence of manganese salts, which doubtless act catalytically.

Copper and Cadmium were next removed by either tube-milling with zinc balls or agitating the neutral liquor from de-ironisation with zinc dust. Copper, and with it arsenic, antimony and bismuth are readily removed, though somewhat greater difficulty was experienced with cadmium. Normally the liquors were then passed forward for electrolysis, but the discovered presence of cobalt introduced a new stage in the purification process.

Precipitation of Cobalt was first and for a

long time very successfully effected by boiling the cobaltiferous liquor with lead dioxide,¹ manganese dioxide, or the mixed oxides constituting cell mud. The reactions presumably take the following course:—



Lead peroxide as a by-product of accumulator manufacture was found to be a suitable and economic material. Usually it contained:—

65 per cent. PbO_2 .

35 per cent. PbSO_4 .

and was used in the proportion of 1 part to 200 parts of liquor. The mixture was gently stirred by agitators or compressed air and the reagent could be used several times before being passed to the lead plant for the recovery of the lead. The cost of materials for this operation was therefore of a trifling character. We were thus able to reduce the cobalt in the liquors down to 5 parts per million, an amount which was, by experience, determined to be quite allowable in the cells. Other oxidising agents such as sodium hypochlorite and bleaching powder were available, but these introduced undesirable constituents, such, for example, as chlorine, into the liquors and had thus to be rigidly avoided. On the works plant it was usual for the head of the electrolytic section to test the "de-cobaltised" liquor before its acceptance.

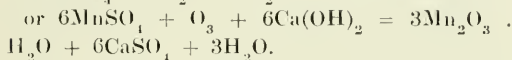
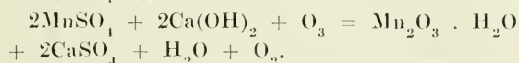
The Removal of Nickel was for some time effected by practically boiling the solution with zinc dust. Again the proportion of reagent was 1 part to 200 of liquor and could be used several times over especially with slight additions of new dust. While this reaction was fairly satisfactory and served for some time, it left something to be desired in that it tended to the production of basic zinc sulphate which slowed down the reaction and also involved loss of zinc. At the same time, the removal of nickel was accompanied by the removal of traces of cadmium which had escaped in tube milling, or on the treatment with zinc fume at lower temperature. It must be stated that temperature was a determining factor in nickel and cadmium elimination.

Use of Ozone.—Ozone or ozonised air is known to be a powerful oxidising agent. It fulfils every requirement of the special conditions attaching to electrolytic zinc sulphate solutions, and may be regarded as one of the cheapest purification agents which can be used. Patent rights have therefore been granted in this and other countries covering the use of ozone or ozonised air.² The particular impurities removed are those which are susceptible to oxidation such as iron, cobalt, arsenic, and antimony. Ozonised air is easily and cheaply produced, readily distributed, and combines both functions of purification and stirring agent. It may here be pointed out

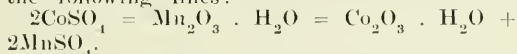
¹ British Patent No. 138,918.

² British Patent No. 138,946.

that oxidised manganese compounds are very effective in the removal of iron from zinc liquors. Normally the crude zinc liquors contain manganese, sometimes as much as .15 per cent. of the metal. This is in the manganous or relatively inactive condition. Even in neutral solution, ozonised air precipitates manganic compounds, but this action proceeds apace in the presence of a little alkali like lime which precipitates the manganese in a form readily susceptible to oxidation. This is effected by ozone when:—



The effect of manganese compounds on iron is too well known to necessitate further reference. With the precipitation of iron, arsenic and antimony come down, in the former case as a basic ferric arsenate. With cobalt compounds the oxidation appears to proceed upon the following lines:—



The cobalt is precipitated as a higher oxide while the manganese in the absence of alkali or alkali carbonate would revert to sulphate. Manganese compounds thus act catalytically, and if any addition is made for this purpose only manganous compounds need be added while other patent specifications covering the addition of manganese compounds specify the more highly oxidised compounds.

There are a number of types of efficient ozone generators available, and the normal product contains 2,000 parts of ozone by volume per million of air. The ozonised air is used at an elevated temperature and serves a most effective purpose in bringing the elimination of cobalt within the scope of the de-ironising operation.

Use of Zinc Dust in Acidified Solutions.—The remainder of the impurities can at once be eliminated by the application of zinc dust to the acidified solution.¹ In a neutral solution the use of zinc dust is attended by the difficulty of the production of basic zinc sulphate. If this type of purification partakes of the nature of electro-chemical action, as indeed it must, it would appear to be considerably facilitated by the addition of a small amount of acid, and this we find to be the case. The liquors to be purified are heated to 96° C., and sufficient acid, either new or regenerated from the cells, is added to bring the acidity up to .1 per cent. Zinc dust is now added in stages and the solution agitated. The amount of blue powder required varies with the quantity of impurities. Normally .2 per cent. suffices. The acid varies with the quantity of zinc dust. An equal weight of 50 per cent. acid is added previous to the addition of zinc dust. In any case an excess of zinc is used, of which an amount equivalent to the acid present and the impurities removed passes into the solution and is subsequently recovered as electrolytic zinc. The sludge containing the impurities such as

copper, cadmium, and nickel may be used again, and where cadmium is present, will in time pay for the costs of the extraction of cadmium.—S. FIELD, A.R.C.S., *Trans. Faraday Society*, February, 1922, p. 418. (H.A.W.)

MANIFOLD USES OF ARSENIC.—The "Engineering and Mining Journal-Press" of New York, in its issue of 20th May, 1922, gives prominence to the uses of arsenic in the cultivation of cotton, particularly in fighting the cotton boll weevil, a pest which, fortunately, we are not at present troubled with in Queensland. Through Government research calcium arsenate has been found effective, when properly applied, against the boll weevil, and as the encroachment of the boll weevil has encompassed the entire cotton-producing section of the United States at this time, the demand for and use of calcium arsenate will be greatly increased. The basic purpose of the poisoning is to merely keep the weevils sufficiently reduced in number so that their feeding will not increase the fruit shed which would be experienced in their absence; that is to say, the weevils are allowed to develop undisturbed until they approach the point of actually reducing the crop, and are then held in check by poisoning just long enough to let the plant set and develop beyond weevil injury all bolls that it will be able to mature.

If calcium arsenate is properly made it will not injure the cotton plants, nor is it nearly so dangerous as paris green, as it does not possess the caustic characteristics of the latter. When properly applied from 5 to 7 lb. of calcium arsenate are required to an acre for each application, the number of applications usually varying from three to five, with an interval of from four to seven days between applications, depending upon the extent of the infestation and atmospheric conditions. Owing to the cost of the material and applying, the Government states that it seems inadvisable to attempt poisoning of land which is not capable of making at least one-half bale of cotton per acre in the absence of weevil injury. The gain secured by poisoning ranges as high as 1,000 lbs. of seed cotton per acre, and on fairly fertile soil, subject to a serious degree of weevil injury, average gains of from 300 to 500 lb. of seed cotton per acre are entirely possible.

The principal difficulty at the present time is in securing an effective method of application. Calcium arsenate is used in exceedingly fine powdered form, and is applied either by machines treating five rows of cotton at a time, or by means of a hand gun, usually carried by a man on horseback, treating one row at a time.

In the United States arsenic is used very largely in the manufacture of insecticides, weed-killers, and cattle and sheep dips. A considerable tonnage of white arsenic is used in the manufacture of weed-killers, which are used principally by the various railroads throughout the country, it being found a cheap and effective method of keeping the roadbeds and rights-of-way free from grass and weeds which otherwise would have to be removed by hand.—*Queensland Government Mining Journal*, July 15th, 1922, page 258. (J.A.W.)

¹ British Patent No. 138,954.

METHOD FOR DETERMINING CARBON MONOXIDE.

—The tannic acid method for the quantitative determination of carbon monoxide in the blood is described in a recent circular (Serial No. 2,356) prepared by Mr. R. R. Sayers (Chief Surgeon) and Mr. W. P. Yant (Assistant Chemist) of the United States Bureau of Mines. Having described the method of procuring blood samples and the process followed in the laboratory for determination tests, they demonstrate the reliability of the process by several tables of results, and express the opinion that the method described is particularly adapted to the requirements for determination of poisoning by CO gas. It can be used for either a quantitative estimation, or, in the absence of a set of standards, it can be used as a qualitative test, the comparison in the latter being made with a single standard immediately prepared from 0.1 c.c. of the blood of an unexposed subject (very conveniently the person making the test). In support of this opinion, the following conclusions are offered: (1) The small quantity of blood needed can easily be obtained without objection on the part of the patient, (2) the solutions used are cheap, common and easily made. The apparatus used is simple and inexpensive, (3) the actual time of making an analysis is not more than three minutes, and results can be obtained in eight to ten minutes; (4) the accuracy is well within the required limits; (5) no great skill or special training is necessary for securing good results; (6) the method automatically corrects for any dissociation of carbon monoxide-hemoglobin due to the dilution with water; (7) the results are more easily obtained, and are as a whole more dependable than with any other method tried.—SAYERS and YANT, *Iron and Coal Trades Review*, June 9, 1922, p. 858. (J.A.W.)

THE STRUCTURE OF COKE.—The formation of high-temperature coke can be divided into three stages: (1) The first stage up to 550 deg. C. is constituted in the initial cell formation, the cells having relatively thick walls of soft material of low specific gravity, the process being accompanied by a swelling of the charge. (2) Between 550 and 850 deg. C. the mass as a whole shrinks, and an increase in specific gravity of the coke material ensues, the resulting coke having thinner but stronger walls and the porosity reaching a maximum. (3) In the last stage the predominant factor is a further large contraction of the mass as a whole, the coke material undergoing no appreciable increase in specific gravity. We are thus left with a strong coke, constituted of small pores with relatively thick walls. At no stage in coking to 1,100 deg. C. does the material approximate to the specific gravity of graphite, 2.3. The authors have followed the process in a fairly typical gas coal, but believe that its essential features would be reproduced in the behaviour of coking coals generally. The swelling of the coal in the first stages of carbonisation is well known to cause difficulties in withdrawing coke made at low temperatures, and in producing sticking at the top of the charge in continuous vertical retorts and gas producers, particularly when fitted with a bell.—H. D. GREENWOOD and J. W. COBB, *Iron and Coal Trades Review*, July 7, 1922, p. 13. (J.A.W.)

METALLURGY

DUST LOSSES DURING MELTING OF GOLD PRECIPITATES.—The losses of gold and silver occurring during the conversion of the precipitate, resulting from the cyanide process, into bullion may occur in two ways: first, there may be mechanical losses during the handling of the precipitate up to the time that it is charged into the melting furnace; and second, there may be losses after the precipitate reaches the melting furnace.

The tests herein described were made on the melting furnaces of the Compania de Real del Monte y Pachuca, in order to determine the approximate dusting and volatilisation losses during the melting of the precipitate and the volatilization loss during air refining of the resulting Doré bullion.

Zinc-dust precipitation is employed at the Loreto and Guerrero mills of the company. The Loreto precipitate averages 75 to 78 per cent. silver and gold and the Guerrero precipitate 80 to 85 per cent. silver and gold. The precipitate as charged to the furnaces contains 30 to 35 per cent. moisture.

The melting equipment consists of two oil-fired reverberatory furnaces with firebrick bottoms and of conventional design. We have designated them throughout this article as the "small furnace," which has a hearth area of 25 square feet, and the "large furnace," which has a hearth area of 49 square feet. Each furnace is equipped with two oil burners which are fired with Mexican crude oil residuum.

The regular procedure in melting the precipitate is to preheat the furnace; then a charge of precipitate, roughly mixed with the minimum amount of flux, consisting of borax glass and finely ground bottle glass, is shovelled by hand from shallow charge cars through the work door at the burner end of the furnace. After the first charge has melted, additional charges are added until the desired amount of bullion has collected on the hearth. The furnace is then tapped and the bullion cast directly into moulds carried on a special casting machine.

In cases where the bullion was air refined, the desired number of charges were melted and the slag was tapped from the top of the bullion, the last of it being skimmed with a rabble and pulled through the work door. After removal of the slag, air refining proceeded. It has not been the practice heretofore to air refine, since there was no advantage in doing this when outside refineries did the final refining, but now that the company is to have its own electrolytic refinery, it has become a matter of considerable importance to ascertain to what extent this cyanide bullion is amenable to air refining. It is generally recognised by refiners that it is desirable to eliminate base metals so far as possible by furnace refining prior to electrolytic refining. The tests of which an account is herein given were made, therefore, not only to determine dusting and volatilization losses but also to determine the extent to which this cyanide bullion could be air-refined.

The low-pressure air employed in the earlier experiments was supplied by a small Buffalo Forge motor-driven fan, while in the later experiments a small Roots blower having a delivery opening of 3½ inches diameter was used.

The low-pressure air was blown over the surface of the bath from a flattened nozzle inserted through an opening in the back of the furnace and directed towards the work door. The resulting molten oxides were drawn off by cutting down the breast at the bottom of the work door. This operation resembles cupellation as carried on in the English type of cupelling furnace. The high-pressure air from the mill compressors, in those tests in which it was used, was introduced through a length of half inch gas pipe inserted through the work door and immersed in the bath of molten metal. This pipe was kept in constant motion so as to reach all parts of the bath. This operation resembles the practice of the copper refiners in eliminating base metals from the Doré bullion resulting from the melting slime from the electrolytic copper-refining tanks, except that they generally use both sodium nitrate and soda as well as air to eliminate selenium and tellurium.

Slag and bullion samples were taken every half hour, the first being taken at the beginning of the refining period. The fineness of the final bullion was determined by the bullion assays on the bars shipped from each melt.

The volatilization losses were estimated from the data secured by filtering a measured volume of the flue gases through paper extraction thimbles, and ascertaining the total volume of the flue gases by Pitot tube readings.

The volume of the stack gases was calculated from Pitot-tube readings made with a standard Pitot tube manufactured by the American Blower Co. and an Ellison differential draft gauge.

Certain general conclusions which are of value may be drawn from the results.

The losses are surprisingly small when it is considered that the precipitate is melted in reverberatory furnaces without briquetting. The average loss of silver is 0.1074 per cent., and of gold is 0.02805 per cent., for the Loreto precipitate during melting. The average loss of silver is 0.0625 per cent., and gold is 0.0150 per cent., for the Guerrero precipitate during melting.

The average loss of silver is 0.0497 per cent., and of gold is 0.0133 per cent., during air refining of Loreto bullion. The average loss of silver is 0.0156 per cent., and of gold is 0.00387 per cent., during air refining of Guerrero bullion. These are average losses for the whole period of air refining which in every case was prolonged more than necessary in order to make sure that the end had been reached. The loss of both silver and gold is considerably greater during the melting of the precipitate and the air refining of the bullion with Loreto precipitate than with Guerrero precipitate. The ratio of silver loss of Loreto to Guerrero for the whole of the precipitate melting period is 1.72 to 1 and the ratio of gold loss is 1.81 to 1. Taking the average hourly losses, the ratio for silver is 1.37 to 1 and for gold is 1.54 to 1.

The ratio of silver loss of Loreto to Guerrero for the whole air-refining period is 3.2 to 1, and the ratio of gold loss is 3.44 to 1. Taking the average hourly losses, the ratio for silver is 4.03 to 1, and for gold is 4.22 to 1.—G. H.

CLEVENGER, F. S. MULOCK and G. W. HARRIS, *Mining and Metallurgy*, January, 1922, p. 11. (H.A.W.)

MISCELLANEOUS.

THE APPLICATION OF STAINLESS STEEL TO MINING WORKS.—At a recent meeting of the Midland Institute of Mining Engineers, Dr. W. H. Hatfield stated that there were many items about a mine which could usefully and economically be manufactured in stainless steel; for instance, boiler fittings, pump rods, pump cylinder liners, valve parts, whether for steam, water, or air, impeller blades, rotary pumps, fan blades, pins for lifting apparatus, hooks, shackles, etc., pins, etc., for coal conveyors; weighing machine parts, *i.e.*, links, levers, etc., gauze for safety lamps, and, although they were costly, wire ropes made from stainless steel had certainly a definite application under certain conditions. In each instance, not only had one to bear in mind the rust-resisting properties, but also the mechanical properties; but since the stainless and rustless steels were now available covering the whole range of mechanical properties, it was merely a matter of selection to ensure that each suggested part had the necessary mechanical properties.

Stainless steel was the name applied to a range of steels which belong to the iron-carbon-chromium alloys, the chromium in all of them ranging from 12 to 14.5 per cent. The percentage of carbon, manganese, silicon and other elements was controlled within desirable limits. As a result of research and works experiment, there were now available other stainless steels than the original one which was produced for cutlery and allied purposes, and which were only rustless in the hard condition. The material was now supplied in a high-tensile and machinable condition, simply required machining to shape, and was satisfactorily rust-resisting without further treatment. There was also a malleable stainless steel which had been evolved for the purpose of supplying a rust-resisting material, chiefly in sheet form, which could be easily deformed cold, hammered, pressed, or drawn into requisite shape, and only required polishing to produce a good surface. There was also the so-called stainless iron, which was a stainless steel in which the carbon content was reduced to a very low percentage. All these steels were manufactured by the electric process of steel manufacture, and required and received the careful subsequent manufacture which was necessary in the case of alloy steels.

The strength of stainless steel, as determined on the tensile testing machine, varies from over 100 tons per square inch breaking load in the hard condition, down to 30 to 40 tons per square inch when fully tempered. The lower figure corresponded to malleable stainless steel or stainless iron. A specimen series of figures obtained with a steel containing 0.30 per cent. of carbon, the test pieces being oil-quenched from 950° C. (1,742° F.) and then tempered, showed that the hardness and tensile strength were substantially maintained up to a temperature of 500° C. (932° F.); higher tempering temperatures then resulted in a thorough

tempering of the material down to very low tensile figures. Results obtained from the Izod impact test showed that as the material was softer, the impact value went up to a very high figure. Apart from the malleable form, stainless steel was almost invariably either used in the hardened and slightly tempered condition, in which it had a tensile strength of about 100 tons per square inch, or it was used in the well-tempered condition, when it had a tensile strength of about 50 tons.

Exhaustive fatigue tests had also been carried out in this steel in both these conditions, with the following interesting results. The tests were performed on the Wohler rotary fatigue-testing machine. The test piece was rotated horizontally at 1,500 r.p.m., with a load hung from one end. This load was changed for each test until a load was reached which just produced the stressing which the material would bear without rupture during 10 million revolutions. The results of the tests on stainless steel, in comparison with those obtained from an ordinary structural carbon steel, showed that mild steel could only be stressed alternately in tension and compression to between 11 and 12 tons per square inch if the material was to survive 10 million revolutions. In the 50-ton (well-tempered) condition, stainless steel would carry alternate tension and compression stresses of 21 tons per square inch, whilst in the 100-ton (knife-temper) condition, this material would stand 32 tons per square inch.

Stainless steel maintained its strength at high temperatures to a much greater degree than ordinary structural steel, and might, therefore, be employed with advantage where parts had to withstand stresses at temperatures above the normal. Moreover, experiments indicated that the energy absorbed in the notched-bar impact test was not substantially modified until temperatures approaching 600°C. were attained. Above that temperature the values increased.

The ordinary methods of preparing steel surfaces should be employed with stainless steel. It should be pointed out, however, that grinding should always be done with water. If by an abrading action the temperature of the surface was unduly raised, the stainless properties were locally affected by the scorching effect thus produced. For the best results the surface should always be as good as could be commercially obtained. Nevertheless, it was not necessary to have a highly-polished surface for the material to be rust-resisting, provided all scale and pit-marks were eliminated.—Dr. W. H. HATFIELD, *South African Engineering* (Midland Institute of Min. Eng.), April, 1922, p. 68. (A.K.)

THE PROTECTION OF METALS FROM HEAT OXIDATION.—The destructive oxidation of metal parts when subjected to the action of high heat is an important factor in the maintenance of these parts in many industrial plants. A recently developed process is known as calorising, and renders the metals highly resistant to the destructive effect of high temperatures.

The process of calorising consists in the formation of a surface alloy of aluminium on ferrous and non ferrous metals, and is achieved by

placing the articles to be heated in an air-tight retort, partly filled with the calorising mixture (which consists of finely divided metallic aluminium suspended in aluminium oxide) and subjecting it to a high temperature for several hours. During the process a current of hydrogen is passed through the retort to ensure an inert atmosphere. Before being placed in the retort, the articles are required to be cleaned and to have a surface free from grease, scale and other foreign matter. This is effected by sand blasting or pickling. The treatment, conducted at high temperature, infuses aluminium into the exposed portions of the metal so as to form a homogenous alloy for a certain depth. This depth ranges from a few thousandths of an inch to the permeation of the entire mass and is governed by varying the duration of the treatment and the composition of the mixture.

In use, the aluminium forms an outer coating of the oxide as the protective surface, and this renews itself should it become injured. Generally speaking, the temperature limits which the calorised surface will withstand varies from 900°C. to 1,000°C. Above this the diffusion of aluminium is so rapid that the alloy becomes too weak in aluminium to form a satisfactory protective oxide.

The protective coating is inclined to be brittle and cannot be bent or hammered cold. It has certain acid-resisting properties and is resistant to furnace gases.—F. S. L. JOHNSON, *The Chemical Age*, July 29, 1922, page 120. (F.W.)

INDUSTRIAL APPLICATION OF THE COLLOID MILL.—The great possibilities afforded by the invention of the Plauson Colloid Mill are explained, in connection with the manufacture of perfumes and essences; also the manufacture of homogenized milk. Its particular application to lubricants, however, is the subject of this abstract.

All kinds of oil, such as mineral oil and coal tar distillates can be converted into stable emulsions. It has been found that under certain conditions aqueous emulsions of oil form pasty masses, some of which contain from two parts of oil as much as one to three parts of water. By adding barium carbonate, magnesium carbonate, magnesium oxide, etc., the percentage of water can be increased still farther without affecting the pasty condition of the lubricant. It is essential that the substances added should be finely divided or colloidal. Graphite, talcum, etc., can be likewise incorporated, forming a homogenous stable paste which possesses valuable lubricating properties, and is completely neutral; no addition of soap is necessary. The colloidal pastes have been found to be more stable than liquid emulsions.

The following are the advantages claimed: They cool the bearings, owing to the high specific heat of water, and are suitable for cylinders of internal combustion engines as the presence of water reduces the temperature of explosion and protects the lubricant from burning. Owing to the absence of alkali, there is no corrosion of the cylinders by mineral matter. It has been estimated that by choosing suitable proportions of oil and water a saving of lubricant up to 60 per cent. can be effected.—S. P. SCHOTZ, *The Chemical Age*, June 17, 1922, p. 790. (F.W.)

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AT

Ordinary General Meeting, 18th November, 1922.

The Ordinary General Meeting of the Society was held in the Assembly Hall, Scientific and Technical Club, 100, Fox Street, Johannesburg, on Saturday, the 18th November, 1922, at 8 p.m., Prof. G. A. Watermeyer (President) in the chair. There were also present:—

27 Members: Messrs. H. R. Adam, C. J. Gray, F. W. Watson, J. Chilton, R. A. Cooper, A. King, H. Pirow, J. M. Thorburn, F. Wartenweiler, John Watson, A. Whitby, H. A. White, J. A. Woodburn (Member of Council), J. A. Boyd, J. M. Dixon, J. E. Homersham, K. Leinberg, L. G. Ray, T. R. Simpson, R. A. H. Flugge-smidt, J. J. R. Smythe, H. R. S. and A. A. Wood.

3 Assistants: Messrs. J. Gibson, R. Kethel, and J. Simpson.

3 Visitors: H. A. G. Jeffreys (Secretary).

MINUTES.

The President announced that the first business of the meeting was to confirm the Minutes of the Ordinary General Meetings held on the 16th September and the 21st October, 1922, published in the *Journals*.

for those months, and which had been issued since their last monthly meeting. This was agreed to.

NEW MEMBERS.

A ballot was taken for the election of new members, and the following gentlemen were declared unanimously elected:—

DURY, C. A. 84, Stiemens Street, Johannesburg: Mine Surveyor.
KNOCKER, P., 1 and 2, Trust Buildings, Fox Street, Johannesburg: Solicitor.
KOTZE, V. F., P.O. Box 103, Crown Mines, Ltd., Johannesburg: Surveyor.
ROSS, G. C., 35, Nelson Road, Booyens, Johannesburg: Miner.
WALLER, F. H. G., 160, Tramway Street, Kenilworth, Johannesburg: Assistant Surveyor.

NEW ASSOCIATE.

The Secretary announced that the following gentleman had been admitted by the Council as an Associate:—

HUNTER, WILLIAM A., King Edward VII. School, Johannesburg: Science Master.

GENERAL BUSINESS.

The President read a cablegram from the Institution of Mining and Metallurgy, London, addressed to the President of the Chemical, Metallurgical and Mining Society, Johannesburg, as follows:—

The Prince of Wales and company, eight hundred, assembled at dinner Guildhall, London, of Institution Mining, Metallurgy and Institution Mining Engineers, send cordial greetings and good wishes to members of profession in South Africa.

CHARLES McDERMID, Secretary.

SOME NOTES ON THE EVOLUTION OF THE KONIMETER.

By R. A. H. FLUGGE-DE SMIDT.

In dealing with the request to sketch the evolution of the konimeter I take it that what is required is an outline of any advances that have been made since Sir

Robert Kotze designed the konimeter that bears his name.

There are three groups of observers using konimeters:—

1. Inspectors of the Government Department of Mines.
2. Inspectors of the Chamber of Mines.
3. Inspectors on each mine.

The Kotze Konimeter.

There had been methods in use in which a microscope was employed to actually count the particles that had been in suspension in a given quantity of air, but it was in the method of trapping for examination that Sir Robert Kotze broke new ground, and after a number of experiments evolved a very practical and efficient little instrument for trapping suspended matter in the air.

This is one of the early types of Kotze konimeter. It has already been fully described by Mr. J. Innes in the March number of the *Journal*, 1918, but perhaps for purposes of comparison with other types the writer may be permitted to repeat a short description.

A pump actuated by releasing a spring, previously compressed, draws 5 cc. of air through a small cell, bounded by a rubber ring. The inlet of this cell is a fine jet tapered down to 0.8 mm. in diameter, through which the air is drawn at a high velocity, causing it to impinge on a cover glass set at a distance of 0.5 mm. from the nozzle of the jet. The outlet is a circular aperture in the centre of which the jet is fixed. The cover glass or slide is coated with a film of some adhesive to which the suspended particles adhere. The cover glass is pressed tightly against the rubber ring by means of a spring clasp. The velocity of the air passing through the jet is about 40 metres per second. The slide is movable, and by a combination of movements in two directions two or three rows of three to four spots each may be taken on each slide. The slides are held in pairs in metal retainers, any number of which may be taken underground. For samples in each different locality a fresh slide must be used.

The Kotze konimeter in use to-day is very much the same as the original model of Sir Robert's. In the newer models the spring is covered in and the clip for holding the slide in position has been improved upon.

One important innovation of Sir Robert Kotze was the attachment of a powerful lens to the konimeter so that by holding the jet in line with the light of an acetylene

lamp the spot could be seen immediately after taking and even estimated with a fair degree of accuracy.

Clifford's Konimeter.

While Sir Robert was evolving his konimeter Mr. E. H. Clifford, to whom Sir Robert had communicated his ideas, was experimenting on similar lines, and produced a konimeter in a different form as shown in this the original model. It is not as compact and handy as the Kotze konimeter. The inlet of the cell is far removed from the exhaust, and the volume of the cell itself is unnecessarily large.

The Circular Konimeter.

The method of changing slides underground and the time spent in locating spots under the microscope were points that obviously called for improvement and stimulated quite a number of enthusiasts to overcome these difficulties.

Cooper's Konimeter.

Mr. R. A. Cooper, at that time on the E.R.P.M., was one of the first to come forward with a konimeter in which a number of spots were recorded on a revolving circular glass disc set in a brass ring which was divided off and numbered. This, the original model, was undoubtedly the forerunner of our present circular konimeter.

Mr. J. A. Woodburn had previously suggested to Mr. J. Boyd the idea of recording a number of spots in a similar manner, while Mr. G. G. Holmes also invented an instrument which was midway between the Kotze and the circular konimeter.

Cooper's konimeter was not very practical, but Mr. Boyd, in collaboration with the instrument maker, Devers, succeeded in evolving a circular konimeter, which, by facilitating routine work, must certainly be looked upon as an improvement on the Kotze model.

The method of trapping the suspended matter is exactly the same as in the Kotze konimeter, except that the exhaust is a little way removed from the inlet. The dimensions of jet and distance between nozzle and cover glass are about the same, but the jet itself is not tapered. The circular disc is much larger than that of Cooper's model, and is divided into 29 sections, on each of which two spots may be taken. The disc is set in a brass ring which is toothed so that by turning a handle it is attached to a worm screw, the other end of

rotated, six half turns corresponding to one division. A strong spring is attached to the lid, which is screwed down over the disc, making an air-tight joint with the large rubber ring. Two apertures in the lid enable one to see the nozzle and read the numbers on the brass ring.

Some circular konimeters are made with a capacity of 5 cc. and others of 10 cc. Varied forms of the circular konimeter have been made for experimental purposes.

(1) *Double Disc Konimeter.*

In this konimeter the exhaust air from one cell is made to pass through a second cell and impinge on a second disc. A certain amount of dust is always caught on the second disc. The general average as determined by a number of samples tested by the Dust Sampling Department of the Chamber of Mines was about 30 per cent. of that caught on the first disc.

If we apply a formula propounded by Dr. J. S. Owens in a recent number of The Proceedings of the Royal Society, we can calculate the actual percentage of dust caught on the first slide. Owens assumes that the proportion of dust caught on the first disc bears the same ratio to the total amount in the air sampled as the amount caught on the second disc does to the contents of the air which escapes through the first cell.

Let t = total number of particles per cc. in the air.

Let C_1 = number of particles per cc. caught on 1st disc.

Let C_2 = number of particles per cc. caught on 2nd disc.

$$\text{Then } \frac{t}{C_1} = \frac{t - C_1}{C_2}$$

Applying this formula to the data collected by Mr. Boyd we obtain:—

- (i) For counts on the 1st slide of under 300, and averaging out 27 results, we obtain the figure of 56 per cent. of the total amount of dust in the sample caught on the 1st slide.
- (ii) For the group of counts ranging between 300 and 700 particles, averaging 23 counts, the result is 44 per cent. caught on the 1st slide.
- (iii) For counts above 700, of which, however, there were only 10, it worked out at 52 per cent. caught on the 1st slide.

Theoretically one would expect a still smaller percentage in the third group, and most likely if a larger number of samples had been taken the percentage would have been lowered.

(2) *Harris' Double Disc Konimeter.*

Previous to the carrying out of those experiments, and as early as 1917, Mr. Lionel Harris had constructed a double disc konimeter. It is modelled on the lines of a Ketzé konimeter, but has a circular attachment. Rubber rings are dispersed with, the contact being brass on aluminium lubricated with vasoline. The capacity is 10 cc. Harris' idea was that by so arranging the sizes of the jets all the particles in counts up to 300 would be caught on the first disc, while for higher counts spots would begin to be visible on the second disc as well, and so form an immediate indicator of bad conditions.

(3) *Multiple Jet Konimeter.*

By constructing a double konimeter with two separate pumps, the one drawing air through a single jet, the other through a group of four jets, simultaneous samples can be taken and comparisons made between the single spot and the group of four spots. In a number of experiments the aggregate for the four spots was always higher than the single spot—the difference being anything from 100 per cent. to as much as 300 per cent. This fact is remarkable, and will be referred to later on. It means that by decreasing the velocity the count is increased.

(4) *Harris' Automatic Recording Konimeter.*

Mention must here be made of a very ingenious instrument invented by Mr. Lionel Harris. This is an automatic recording konimeter, actuated by clockwork that can be set to work over different periods. The spots are recorded spirally on a glass disc. The principal idea of this instrument was to be able to sample air in places where it would be dangerous for human beings to linger, as, for instance, a development end immediately after blasting.

With a view to eliminating certain objections to the circular konimeter the writer has made a few minor improvements.

(1) By means of an inner rubber ring with a V-shaped inlet the volume of air between the floor and cover of the cell was greatly minimised. The circular portion is

partly filled in with white enamel to further reduce this volume.

(2) By marking the glass disc itself. The space between two concentric circles which contain the path of the nozzle is divided into sectors which correspond to those on the brass ring and are also numbered. These division lines are easily visible while using the instrument underground, and enable one immediately to see whether the glass disc has become detached from the brass ring, while under the microscope it becomes impossible to make a mistake in locating spots.

(3) Spare discs in metal retainers are very useful in case of accidents, or when a very large number of samples need to be taken.

(4) By attaching a rubber tube to the inlet of the konimeter and drawing the air out of a measure glass held over a basin of water the pump is easily calibrated and leakages detected.

Microscopic Work.

Since Mr. Innes' paper in 1918 the advance in microscopic work is fairly well known.

Dr. Rogers has described to us a method of distinguishing silica from all other particles, and Mr. Atkin has shown us how easy the process is, while Mr. Gray has drawn attention to methods for eliminating CaSO_4 and other salts, but we are still far from certain as to how to differentiate between injurious and non-injurious particles. Experiments carried out by Mr. Boyd leave a big doubt as to whether all the particles of CaSO_4 are eliminated by treatment with HCL fumes. CaSO_4 is only soluble to a limited extent in a solution of HCL , a matter of 4 per cent. Slides prepared for acid treatment with a water-soluble medium must be freshly prepared before using, and must be treated and counted soon after using, preferably on the same day.

Speaking generally, a standard of 300 particles per cc. as counted with a light background compares with about 500 counted with a dark background. The Chamber of Mines still adhere to the former method for their routine work, while the Mines Department have adopted the latter.

The finer points of microscopy I leave to those more competent to deal with them.

COMPARISONS BETWEEN KOTZE AND CIRCULAR KONIMETER.

(a) *The Kotze Konimeter.*

The main disadvantages of the Kotze konimeter are:—

- (1) The necessity for frequently changing slides in a dust-laden atmosphere.
- (2) The time spent in locating spots under the microscope.
- (3) The time spent in cleaning and preparing a large number of slides.

(b) *The Circular Konimeter.*

The circular konimeter was devised to eliminate the main drawbacks to the Kotze konimeter.

- (1) For ordinary routine work one circular slide is sufficient for the number of samples to be taken.
- (2) The time spent in locating spots is reduced to a minimum.

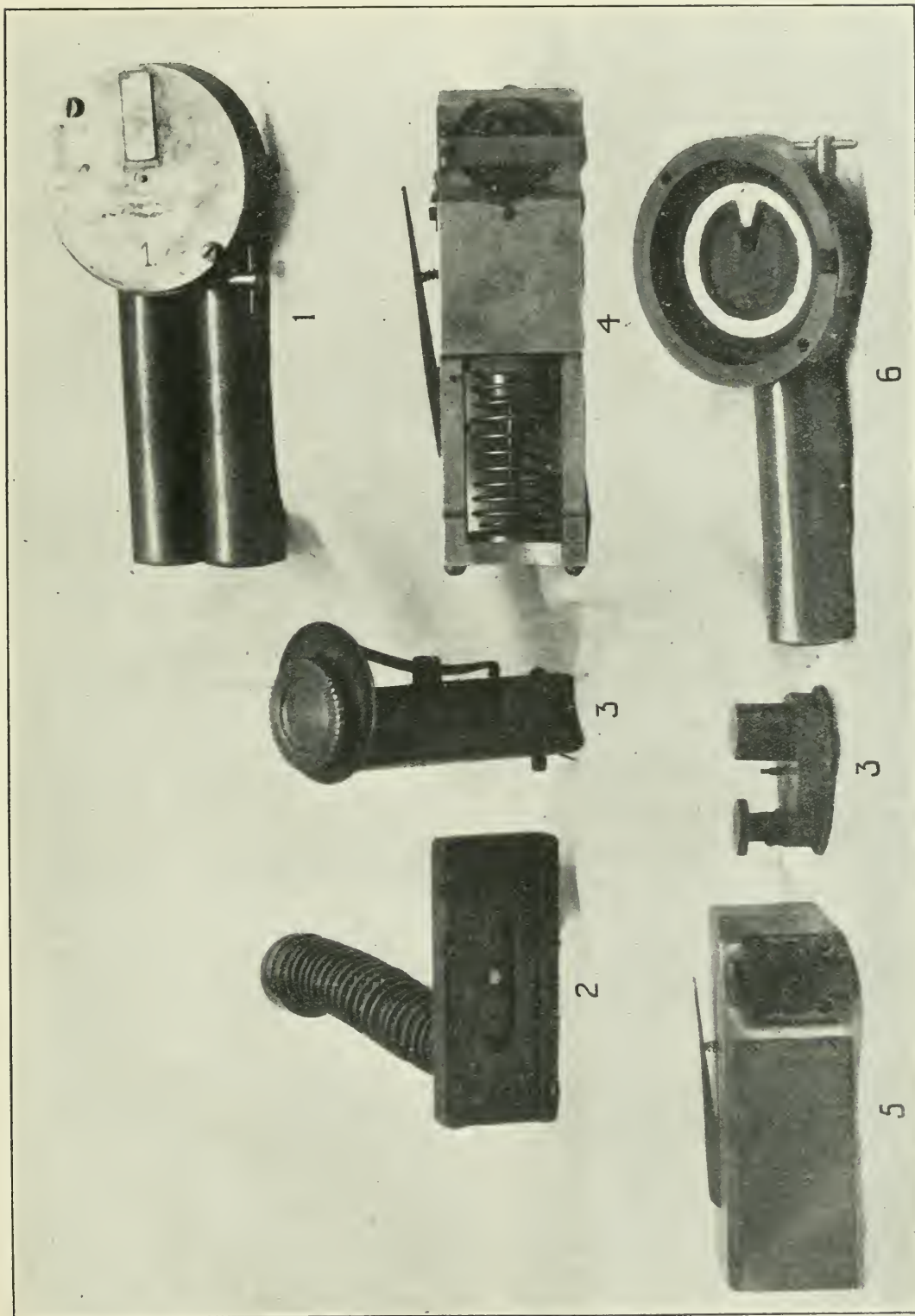
In practice when commencing operations with a circular konimeter No. 29 is placed opposite the nozzle, and instead of one puff ten or more are taken so as to produce a dense spot. This so-called clearance spot is visible to the naked eye, and as such easily placed in position on the stage of the microscope. A simple adaption for the microscope with fine screw adjustment enables one to accurately centre the clearance spot, after which one automatically advances from spot to spot by turning a handle attached to a worm-screw identical with that in the konimeter.

To enable one to treat spots on a circular slide with acid the writer uses a thin glass disc set in a leaden frame in place of the watch-glass. The blotting paper is placed in the centre, and all the spots are treated simultaneously, while the disc can be rotated in the usual manner.

If there is any advantage that the Kotze konimeter has over the circular konimeter it is unknown to me.

The Kotze konimeter is used exclusively by the Mines Department. One can understand that once having been installed the matter of additional expense has to be well justified, and that the advantages of the circular over the Kotze konimeter would only be appreciated where continuous konimeter work is being done.

To sum up the present position, the konimeter is a very excellent instrument for obtaining comparative results, particularly by one man or one mine, and forms a very



1. Multiple Jet Konimeter.

2. Clifford's Konimeter.

3. Cooper's Konimeter.

4. Harris's Double Disc Konimeter.

5. Improved Kotze Konimeter.

6. Improved Circular Konimeter.

good meter of improved conditions, but it is hopeless to try and compare the results obtained by the Chamber of Mines Dust Sampling Committee, using as they do for all routine work vaseline on the slide and counting with light background illumination, as against those obtained by the Mines Department using only dark background illumination, as well as a different adhesive.

All instruments and methods would have to be standardised before such comparisons could be made, and a limit be fixed of so many particles per cc., above which conditions would be considered bad.

As regards the instrument itself the two main difficulties would be:—

- (1) Strength of spring in the pump.
- (2) Adhesive to be used on slide.

All other points, such as size of nozzle, capacity, etc., are easily standardised.

We will consider (1) and (2) together.

According to Dr. Owens the velocity of the air in the jet is of the greatest importance. Owens, using a plain glass slide with no adhesive, found that the counts increased with the velocity, while in the experiments with the multiple jet we have seen that using vaseline on the slide the reverse was apparently the case, that is, the lower velocity gave the higher count. This can only be explained by the fact that however thin we may apply the vaseline it is still microscopically speaking of considerable thickness, and by increasing the velocity a larger number of particles become completely imbedded in the vaseline.

Does not this suggest extraordinary variations affected by the personal factor in applying any adhesive?

The writer has personally obtained very well-defined spots on a glass disc that was free of any adhesive, and the counts compared very favourably with check spots taken and counted by Mr. Joseph, who was using a Kotze konimeter.

Dr. Owens, by using a velocity of 90 metres per second, obtained an efficiency of over 80 per cent., which would suit us admirably. It seems feasible to me that we should adopt a similar practice, and so eliminate one of the difficulties of standardisation.

The spring of the pump could be calibrated from time to time and easily replaced.

Standardising microscope work presents no difficulties except that each part of the microscope as well as strength and method

of illumination would have to be standardised.

The U.S. Bureau of Mines published the results of some experiments, and claimed that, as a method of trapping, the sugar-tube caught 13 times as much dust as a circular konimeter—the result in both cases being referred to the number of particles per cc. using a magnification of 200.

On analysing the figures the statement does not appear so startling, and applies more to enormous counts up to 12,000.

For a low count the konimeter actually caught more than the sugar-tube, while for a sugar-tube count of 1,000 the konimeter registered 400, comparing more or less with the percentage obtained in the experiments with the double disc konimeter. For counts much above 1,000 the konimeter no longer functioned.

On these fields the gravimetric method has already done the heavy work, the konimeter is only called upon to do the finer work, and it is well qualified to do so.

For information concerning the work done in the Mines Department I am indebted to Mr. E. H. A. Joseph, Assistant Inspector of Mines, Brakpan.

For the loan of most of the models which are to be described and for much assistance and information regarding the work done by the Chamber of Mines Dust Sampling Dept., I am greatly indebted to Mr. J. Boyd. The work done by individual Mine Inspectors is not co-ordinated and data is very difficult to obtain.

Mr. C. J. Gray said he had listened with great interest to Mr. de Smidt's description of the evolution of the konimeter, in which he had dealt with its various forms and the improvements or otherwise which had been made.

A number of men who had to deal with the instrument thought they could improve it in some particular way. Unfortunately, some of those, although working with the best of intentions, had failed to realise the underlying principles of the original konimeter. That konimeter was not the result of a simple idea only, but of the most careful working out of the idea by Sir Robert Kotze, who prescribed, for instance, the proper size of nozzle or jet and the proper speed at which the air current should impinge. In the circular konimeter there was a comparatively large air space between the cylinder and the jet; the result was

that there was retardation of the velocity of the air and impingement. That made a radical difference in the working of the instrument. Though a certain amount of convenience was obtained the instrument was inaccurate in comparison with the original instrument. The author had given a great deal of attention to the matter, as they could judge from his remarks, and he had removed or greatly lessened a number of the objections to the original circular konimeter. Without examining his instrument or using it one could not say whether he had now made an instrument which was equal in accuracy and reliability of results to the original Kotze, although probably he had approached it. There were, however, one or two difficulties which struck one as still remaining; for instance, with a large circular disc they had a large bearing surface between the glass and the rubber ring. Now, one of the difficulties even with the Kotze konimeter was the danger of leakage between the glass and the rubber ring. If there was the slightest leakage there they would have a serious difference in the result obtained—the air got in around the slide instead of through the nozzle. With the large ring in the circular konimeter it was far more difficult to make everything air-tight, or so he understood it. Another practical difficulty was that, in order to get good results there must be an exceedingly thin film of adhesive on the slide; they must have, even with vaseline, an exceedingly thin film. If it were thick the dust spots were inclined to run and were indefinite. With the new process, using glycerine jelly or fish-glue, still thinner and more even film was needed, as otherwise, under the acid treatment, with absorption of moisture in the film, the spot ran. It must be far more difficult to spread the adhesive evenly over the comparatively large circular slide than it was over the small microscope slide used with the Kotze instrument.

In spite of those difficulties one must realise there were conveniences in the circular konimeter. It was certainly a more convenient instrument to hold in the hand, and it had a distinct advantage in making it unnecessary to handle the slides in putting them in and taking them out in the mine. If one were careful he could do that perfectly well with the Kotze konimeter, and it was done regularly. Men handled large numbers of slides with that konimeter, with

very few spoilt through handling; but it required a certain amount of care, and one would naturally prefer the circular konimeter for that reason, if it gave equally accurate, definite results. There was, however, another point in connection with it. They knew the position with regard to cameras, there was the roll-film camera and the plate camera, and some cameras using either plates or films; he happened to possess one of that kind. In practice he generally used the plate, as he generally wished to take only a few photographs at a time; he did not wish to use a whole roll-film, and he found the plate more reliable than the film. If he were going on a tour and wished to take a large number of photographs he might sacrifice the quality of his results for the convenience of using a film with which he could take a number without the trouble of changing plates. The circular konimeter resembled the film camera. He did not think it was always an advantage. There was not very much in it as far as the Inspectors of Mines were concerned. Inspectors counted their own spots; they took the slides, examined them, and promptly counted the spots themselves. They saw exactly what they were dealing with. The practice, on the other hand, in the Chamber of Mines Dust Sampling Department, where the circular konimeter was used, was that the spots were not counted by the man who took the samples; he sent them in much in the same way that a sampler sent the samples to the assay office.

Reference had been made to a test as to the amount of dust which was caught by the konimeter on the first slide. It should be clearly understood that those were not the results obtained with the Kotze konimeter as originally designed, and they did not show what the position was with the Kotze konimeter. Tests had been made with the Kotze konimeter, and they showed that it caught a larger percentage under ordinary conditions than the figures given would indicate.

Reference had also been made by the author to some investigations by Mr. Boyd as to the effect of calcium sulphate. He thought it would be recognised by those who worked with the glycerine jelly or fish-glue and acid process for eliminating calcium sulphate, that if they had a large quantity of calcium sulphate and caused that to adhere to a slide with little fish-glue or glycerine jelly, they could not expect an acid vapour

to dissolve it; but in practice they were dealing with small quantities. The silica particles which it was considered caused phthisis, and the associated particles of calcium sulphate were about one micron in diameter. One micron was $1/25,000$ of an inch. If they compared that with the thickness of the thinnest film they could practically make with a material like glycerine jelly, and considered the amount of moisture and acid that film would absorb, they would not expect difficulty in dissolving the amount of calcium sulphate in ordinary spots. One had to look at the supposed difficulty with a sense of proportion.

Although he had not fully agreed with the author on some points, he certainly felt they were indebted to him for raising the subject and for the interest he had shown in it. Too often, when dealing with konimeter work and sugar-tube work, people jumped to conclusions; they gave no reasons, simply vague assertions. The author had certainly thought about the matter and had given them the result of his thoughts. They were indebted to him for that, and he therefore proposed a hearty vote of thanks to Mr. de Smidt for his paper.

Mr. H. Pirow, in seconding the vote of thanks to Mr. de Smidt, said he would like to refer particularly to one point, that was in connection with the preparation of the slides. They had found when working with glycerine jelly and fish-glue it was necessary to have an extremely thin film on the slide, that was particularly the case with glycerine jelly. They found, by using the glass rod, they got a thin film, but that putting on the acid it sometimes caused the spot to run. On investigation it was found to be mostly due to a thicker amount of glycerine jelly being at the edges of the slide. They obviated that by carefully wiping the slide all round the spots. That, he thought, was a distinct advantage of the Kotze konimeter—that, in preparing the slides, as Mr. Gray had emphasised, they undoubtedly got a much thinner film than one would on the big circular disc.

The author had mentioned that the changing of slides underground was done in a dusty atmosphere. Personally he had found, and he thought other officers who had used the instrument had also found, that the area seen under the microscope all round the spots was practically free from

contamination. So that point did not appear to be a serious disadvantage of the Kotze konimeter.

There was also the matter of the number of spots one could take with the Kotze konimeter; and here, again, he did not see any disadvantage in it, because with 12 slides, which was the ordinary number one took underground, one could take at least 144 spots, which would compare quite favourably with the 60 odd spots of the circular konimeter.

The author had mentioned that Sir Robert Kotze had lately improved on the konimeter by putting on a small microscope by means of which one could immediately estimate the amount of dust underground. He (the speaker) had been able to use that microscope to a certain extent, and in tests which had been made they had found the estimate underground to be within 30 per cent. of the count one would get on the surface under the microscope. That instrument had appeared to be particularly useful in impressing the fact that invisible dust was being inhaled on miners and mine officials underground. It was quite a simple matter—he had the instrument there—for the miner to take his own spot; as a matter of fact one could go so far as to put in the slide, draw back the spring, and while looking through the instrument at the light depress the spring and actually see the dust particles impinging on the glycerine jelly. It proved very popular on the mines, and aroused a great amount of interest.

Another little improvement which had since been made on that particular instrument was a small spring attached to the pivot of the clip holding the slide; by means of that spring, which could be easily pressed down with the finger, the slide may be moved in any position under the small microscope without taking the slide out. That, of course, underground, where samples were taken under difficulties, appeared to be of great advantage. He had much pleasure in seconding the vote of thanks to Mr. de Smidt. He might mention that it would be quite easy for any of the members present to take a sample there by means of the instrument, and members could see for themselves how simple a matter it was.

Mr. A. J. R. Atkin thought that Mr. de Smidt was to be congratulated on having got together such a large collection of instruments of various designs all embodying the principle discovered or applied by Sir

Robert Kotze in the invention of the original konimeter.

Movement of the spots when they endeavoured to dissolve the calcium sulphate by means of hydrochloric acid was generally due to the film of adhesive being uneven in depth. It was difficult to get slides which were actually flat. If they pressed them with a rod they would find very many of them were uneven on the surface, and the hollow, where the adhesive was considerably thicker than they required it, would invariably float a spot after it had been enclosed for 10 minutes or so in the hydrochloric acid cell.

As regards the working of the instrument, he must have taken a good many thousand samples by now, and he saw no reason to depart from the original design of the konimeter; in fact, the first one made in metal he still used every day. He would ask Sir Robert Kotze's permission to let him explain to them the evolution of the design of the Kotze konimeter, because, as far as he knew, it had never been published.

Mr. R. A. H. Flugge-de Smidt said there were just one or two points he would like to reply to that night.

First of all, Mr. Gray had referred to some experiments which Mr. Boyd had been carrying out with reference to calcium sulphate. Mr. Boyd tried to obtain the same conditions as they would get in connection with an ordinary dust spot—that was, to form a little fog of solutions carrying calcium sulphate and take a spot of that fog. Viewed under the microscope it was extraordinarily like an ordinary Leyner-fog spot. He took a micro-photograph of that spot, then treated it with acid, and took another micro-photograph; and it was not as though only the large particles were left; a very large number of the tiniest particles were left. It would seem that that more or less corresponded with the conditions they had underground, if they had a lot of calcium sulphate in their mine water.

Then, as regards the applying of the adhesives in connection with the Kotze konimeter; a large number of slides had to be prepared and one slide was as uneven as another. Why should they have greater varying thicknesses on the single slide of the circular konimeter?

He must say he had been rather diffident about reading that paper except for this particular reason, that he had nothing to

do with the Chamber of Mines or the Mines Department; and so took an entirely independent view.

The number of samples depended on their routine work. Personally, he found that in studying conditions underground it was useless, or next to useless, to go in and take one or two samples in one place, and then go on to another. He very often in a single place took twenty gravimetric samples and forty konimeter samples to correspond—that meant a considerable amount of counting—and under such conditions he very much appreciated the advantages of the circular konimeter.

As regarded the figures for percentage of total amount. He granted them this, that it was certainly the circular konimeter he was referring to, not the Kotze konimeter. It was quite likely they would get a Kotze konimeter to give them 70 per cent. of their total particles; but, with certain improvements on the circular konimeter, they could probably get the same percentage. After all, they were not out to get 100 per cent. efficiency; that was not at all necessary. If they got a certain percentage, so long as they knew they were getting that percentage more or less, and made that the standard all over the Reef, it was quite sufficient.

The microscope attachment described by Mr. Pirow was equally applicable to the circular konimeter, and it had been used on the circular konimeter. More of them would be attached to circular konimeters were it not for the difficulty of obtaining the lenses.

With reference to the volume under the glass disc in the circular konimeter it was quite true that this volume had a certain influence on the action of the spring. The first effect was to cause a partial vacuum, and this supposedly retarded the action of the air entering the jet. The speaker was of the opinion that this need not necessarily make such a difference to the actual spot. He had put the inner rubber ring into his konimeter simply to obviate what he called a more or less theoretical disadvantage of the circular konimeter, not that he thought there was any great practical advantage in it.

Mr. H. A. White thought it would be an advantage to the Society if Mr. Flugge-de Smidt could obtain those micro-photographs of the experiments with calcium sulphate to which he had referred, for reproduction in the *Journal*.

Mr. C. J. Gray asked Mr. de Smidt if he would tell members what medium was used by Mr. Boyd in his tests and what was the count he had obtained in using the artificial fogging and calcium sulphate. The medium and the amount of calcium sulphate would affect the matter.

Mr. R. A. H. Flugge-de Smidt, in reply to Mr. Gray, said unfortunately he could not give the figures with regard to Mr. Boyd's count. He had simply shown him (the speaker) the micro-photographs. He himself was convinced just from glancing at the micro-photographs that there was not such a very great difference as one would expect; but he would not like to say what the actual count was. He did not know whether it was a 5 cc. or a 10 cc. konimeter; at any rate, it was not a very huge count; he would say it was under 500. Some particles were undoubtedly large.

The medium used, he believed, was fish-glue on the slide, which was afterwards treated with hydrochloric acid.

In reply to Mr. White, he would ask Mr. Boyd's permission, and, he had no doubt, he would let them have the micro-photographs referred to.

SYMPOSIUM: MINERS' PHTHISIS.

Mr. J. A. Woodburn said he would like to make a few remarks which would probably come under the heading of the Symposium.

Two or three days ago a paper read before the Broken Hill Branch of the Australasian Institute of Mining and Metallurgy had been sent to him through our Secretary. It was a paper on Ventilation and the Kata Thermometer, and it showed that mining men in other parts of the world were taking a keen interest in the subject. A number of very interesting figures had been published in connection with the paper. One or two points in connection with it might be of interest to members here, more particularly after they had heard Mr. C. J. Gray's remarks on the results of the Committee which had been investigating Miners' Phthisis, or Ventilation and Dust during the past two years.

There was one point at the beginning of the paper which he thought worth reading:

"That atmospheric conditions have an important bearing on health is generally recognised, but the effect on

"efficiency has not always been considered. . . . This fact was fully recognised by the engineers employed in designing the Ford motor-car factory. "Ventilation was the first consideration, "and it might well be claimed by the "author of the article in the *Engineering Magazine* for November, 1914, "that the factory was equipped with "the best-built ventilation system ever "installed in any factory building."

Then:

"If ventilation affects the efficiency of "the surface worker, it is quite certain that the underground worker is "affected in a similar manner, and "probably to a greater extent."

As one who had many years ago expressed the view that efficient ventilation throughout the mines would go a very long way in reducing miners' phthisis, probably he was more inclined to seize any information which tended to bear out that opinion, and he could not emphasise it sufficiently.

There were two pages in the paper illustrating various levels in a mine 1,200ft. deep, with levels at the various stages: 1,100, 900, 800, 500, and 400. The air was evidently taken down the shaft to the bottom level first. The temperatures were given, the velocities were measured, and the dry Kata and wet Kata were given in a table. From those there were some interesting conclusions to be drawn. At the bottom level, to which the fresh air was conducted, the temperature ran to about 70°, both wet and dry, so that the air was almost saturated. The velocity was given in one case as 65ft. and in another as 35ft. per minute. The Kata reading was 5.5 dry and 14.6 wet. The context stated that work was carried on there under fairly favourable conditions, although the Kata reading was very low. As the air rose through the various levels it became heated, so that at the higher levels the temperatures were higher than at the lower levels. Evidently there was not much moisture, because the humidity decreased with the rise in temperature and the difference between the wet bulb and the dry bulb became greater at the 800ft. level. In one section where there was still air the temperature of the wet was 76° and the dry 81°, the humidity 76%; the dry Kata was 2.7 and the wet Kata 9.6. In another section of the same level where the velocity was 152ft. per minute, but practically the same temperatures, wet and

dry, the dry Kata was 6.7 and the wet 20.8, which was an eminently satisfactory result, and was entirely due to the 152ft. velocity. The low Kata reading at the bottom level with 35ft. velocity was evidently due to the humidity of the atmosphere. Quoting again:

"The writer would particularly draw attention to reading No. 20, which, compared with the readings obtained in the stopes on the 970ft. level, clearly indicates the effect of air-movement. Although the temperature in this stope is 80½° F. dry bulb and 75½° wet bulb, the cooling power of the air is 6.7 dry and 20.8 wet, as against 5.4 dry and 14.9 wet at the downcast shaft."

That was, at the bottom level.

Another point of interest was using the Kata thermometer as an indicator of velocity. They would probably remember that some two years ago when a paper was read before them on the Kata thermometer it was claimed that the Kata thermometer could be taken not only as equivalent to the anemometer but probably superior to it, in the estimation of air velocity. The paper read before the Broken Hill Branch, as the result of various readings, distinctly combats this claim. Quoting once again:

"Whilst advocating the use of the Kata, the writer would not discard altogether the thermometer. The thermometer, however, does not always indicate the true atmospheric conditions and degrees of comfort, particularly when the temperature of the air is 75° F., or higher, as reference to reading No. 20 and comparing it with reading No. 19 will show. The wet and dry bulb readings are practically the same, but the cooling power of the air in one case is two and a half times that of the other. Consequently working conditions are better where the velocity of the air is greater. Some experiments have been conducted with a view to using the Kata as an anemometer. It has been found, where the current of air approaches 100ft. per min., or when eddy currents are not present, that the Kata reading can be relied upon as a means of calculating the velocity, using the formula as supplied by Dr. Hill. When eddy currents are present, however, Kata readings cannot be checked against the readings obtained when the

"usual mine methods of measuring slow velocities are used."

He gives one or two instances where the air was measured by putting a strong scent into the atmosphere, recording when it reached a certain stage and measuring the air in that way as against the result calculated from the Kata.

There was another point which he thought worth mentioning. The author said:—

"Apart from improving working conditions in development ends, the supply of an additional quantity of fresh air has the effect of reducing the dust content, and this in itself is a most important consideration."

He had quoted these extracts because during the past two or three years they had had papers not only in this Society, but also in other Societies, on that very important subject, and originally it was claimed that an increase of air would increase the dust; that was practically accepted by the majority.

They had been listening to the evolution of the konimeter that night. The point had occurred to him that, had the konimeter been in existence ten or fifteen years ago, when the sugar-tube method of dust-sampling was started, the konimeter would probably have been so fogged that the benefit which they now derived therefrom would not have been so apparent. The sugar-tube method at least eliminated a large percentage of the dust, and, after it was thought the air was practically clear, the konimeter appeared and indicated that even in that clear atmosphere there was still a considerable amount of harmful dust.

He did not think it was at all possible to establish any standard by means of the konimeter to define the limits of a suitable working atmosphere, and he was still strongly of opinion that, if as much ingenuity had been used in the past ten or fifteen years in conducting air throughout the mines and improving the conditions with regard to ventilation, they would have reached a much more satisfactory condition than they had so far done. In order to illustrate that he would just like to mention one case in connection with an outside mine where the reef was lying practically flat and was only about 18 to 20 inches wide, and the roads, as they could imagine, were not too high. There was no artificial ventilation. In one section of the mine two years ago, when it was taken over by a new manager, stopping was taking place about 1,500 or

1,800 feet in, and there was only one ventilation shaft about half-way in. This section had been developed with only one drive, and the old stope faces and the old drives had become clogged up, so that only a very small number of boys could work. A fan was installed in the drive with several hundred feet of piping, and this was taking air in to the few boys who were working. Afterwards, by improving the connection between the main drive and the stope, the stopes were now probably 150ft. further in, and the development drives were between 200 and 300ft. still further in. The air was passing through the drives at such a rate that it deflected the flame of a candle, and one could see the smoke from the candle going in a particular direction. Of course it was guided by screens, any short-circuiting was stopped, and a certain amount of intelligence was used to guide it. The old fan which was in use was discarded. He wished to emphasise that it was not so much the getting of a fan on the surface or underground to blow a large quantity of air in any particular direction, but that the guiding of the air and the laying out of the workings so that the air would travel to the place where it was required was essential. This must be emphasised, and emphasised continuously, until better ventilation throughout the mines was obtained.

NOTES ON THE MANIPULATION OF OSMIRIDIUM CONCENTRATES.

By R. A. COOPER.

(*Printed in Journal, February, 1922.*)

REPLY TO DISCUSSION.

I think that it was only to be expected that very little discussion would be forthcoming on this subject as only a very limited number of people have yet had an opportunity of doing much chemical work on osmiridium concentrates. However, I hope that anyone encountering either unusual occurrences of the platinum group metals, or fresh difficulties in the treatment of Rand concentrates, will reopen the subject before this Society and so give us another opportunity for discussion and some increase in our fund of available information.

Mr. White, in his discussion of Mr. J. R. Thurlow's note on the metallurgy of

osmiridium, suggested that the present methods of concentration fail to collect anything like the whole of the platinum metals present in the ore. It is almost impossible to determine the amount of these metals accurately in reef samples, so the point is somewhat difficult to prove, but the difficulty of recovering osmiridium from heavy concentrates by such mechanical means as panning points to the probability of Mr. White's idea being correct. The osmiridium is not easily wetted, and usually floats with ease if it happens to come into contact with the surface of water. Then also the widely varying specific gravities of the metals must tend to separate them; the natural alloy of osmium and iridium has a specific gravity of about 20, while the slightly impure platinum is almost the same, and gold is very little lighter. Ruthenium, rhodium, and palladium have gravities of only about 12. Ruthenium is usually present in concentrates to the extent of about 10 per cent., but rhodium and palladium are present only as traces or not at all. It may be significant that a furnace by-product has recently been examined which contains a considerable quantity of platinum group metals, of which ruthenium forms about 50 per cent.

I think that the precious metals are naturally present in the reef as the alloys platinum-silver soluble in nitric acid, gold-silver soluble in dilute aqua regia, platinum-rhodium soluble in concentrated aqua regia, ruthenium soluble in fused potash and potassium nitrate, and osmium-iridium. The latter, under the microscope, appears to be unaffected by these solvents.

THE ENDLESS ROPE HAULAGE AT THE MODDER EAST.

By J. CHILTON.

(*Printed in Journal, April, 1922.*)

REPLY TO DISCUSSION.

When this paper was introduced Prof. G. A. Watermeyer expressed a desire for more information in regard to costs. Since the industrial upheaval old methods and standards have been greatly altered, and the process of cost reduction is still going on, so that any figures given now on the cost of endless rope haulage would probably be misleading. Only under more settled conditions would an analysis of costs have any value.

The haulage has now been running over twelve months, and its flexibility becomes more and more apparent, and although, as Lieut.-Colonel Pam suggests, it is a reversion to old coal mining practice, yet the merits of the chain grip attachment have not yet been fully recognised in this country.

NOTES ON OSMIRIDIUM.

By J. R. THURLOW.

(Printed in *Journal*, May, 1922.)

REPLY TO DISCUSSION.

As there was no discussion on these notes there is nothing to reply to. I think, however, they have served a useful purpose in arousing interest, as I have had many personal inquiries regarding minor details on the subject. In the meantime several operators have utilised the method to test concentrated products for the rarer metals, with varying results.

In general it may be noted that usually some evidence was obtained of the presence of osmiridium, although not necessarily in sufficient quantities to justify its systematic recovery.

A recent development of technique is to take advantage of the Wilfley or other concentrating machine, which now forms part of most of our up-to-date clean-up departments, and to separate the extreme edge of the concentrate streak just as it leaves the machine by brushing it into a receptacle and re-dressing it in a panning dish.

Mr. H. R. Adams (who had read Mr. Thurlow's reply to discussion, in the unavoidable absence of the author) said, perhaps he might be permitted to mention that Mr. Thurlow's paper had been reprinted in the *Chemical News*.

TECHNICAL EXAMINATION OF LUBRICATING OIL AND GREASE.

By F. W. WATSON and H. D. BELL.

(Printed in *Journal*, May, 1922.)

REPLY TO DISCUSSION.

The authors are disappointed that there has been practically no discussion on this paper. We desire to thank Mr. Adam for

his remarks on the night the paper was read. With regard to the specification requirements, Mr. Adam seemed to consider that a more scientific method than the fact that lubricants satisfying these conditions gave good use in practice, should be employed in drawing up the specifications. So far we know of no other method, but, after all, the result of long practical experience seems to the authors to be a satisfactory basis to work upon.

Another point mentioned by Mr. Adam was the determination of volatility. We are afraid that a distillation method would present considerable difficulty, as it is not analogous to determining water in a grease where xylol is used boiling about 40° C. above water. In a volatility test a constant temperature of 370° F. has to be maintained which would be exceedingly difficult if a distillation method were attempted.

The authors still hope that a paper dealing with physical tests will be forthcoming in the near future.

GOLD PRECIPITATION BY ZINC DUST AND IN CONJUNCTION WITH THE DE-AERATION OF SOLUTION AT MODDERFONTEIN "B."

By S. NEWTON and L. L. FEWSTER.

(Printed in *Journal*, June, 1922.)

FINAL DISCUSSION.

Mr. F. Wartenweiler: The paper, although remarkable for its wealth of detail, is silent on the matter of cost of the operations described. By permission of the Central Mining and Investment Company, Ltd., I am able to submit some information which I have collected on this point. In the following tabulation the precipitation process costs are divided according to the steps involved in recovering the precious metal from its solution.

	Cost per fine ounce gold precipitated. In pence.
Precipitation	4.972
Clean-up and acid treatment	0.343
Smelting and cupelling ...	2.446
Casting to bars	0.471
Supervision	0.475
Total	8.707

On a plant using filiform zinc precipitation under similar conditions, excepting grade of ore, the cost per ounce is 10.11 pence. The zinc dust consumption on the one plant is 0.247 lb. per ounce gold, while on the plant using filiform zinc it is 0.650 lb., indicating the greater efficiency in the utilisation of the precipitant. I use the basis of cost per ounce gold recovered for the reason that the amount of zinc used is largely a function of the gold content of the solution, and because other bases are affected to even a greater degree by local conditions. The cost of producing a unit of the precious metal product is inclined to be lost sight of in the stereotyped "cost per ton milled" basis.

A feature of the zinc dust precipitation process, which the authors did not emphasise as much as warranted, is the advantage of the complete clean-up. The desirability of recovering all the gold at the end of each month or whenever the presses are emptied of precipitate, is being recognised to a greater extent on this field. Especially as more of the gold recovery responsibility is being placed on the cyanidation section so does this feature become more important for technical and commercial reasons. With amalgamation eliminated, the complete clean-up may be said to be a necessity. The precision made possible with such a clean-up is not only of use to the metallurgist, but should give comfort to the management as well. The loss of interest on gold retained in extractor boxes is a hackneyed point, but will bear repetition.

Undoubtedly the process will experience further development as time passes. The uncertainty of the quality of the zinc dust supplied has often given cause for anxiety, and one is impressed with the advantage of securing a local supply. It is possible to produce a clean zinc dust from spelter and eliminate the risk of oxidation and cementation by breakage of containers and sea water incurred during transport. Zinc dust paper, containing 60 per cent. zinc by weight, has been made by the writer with the object of supplying a definite mass of zinc through which the solution must pass, but facilities for its trial have not yet become available.

Quite recently an improvement on the triangular precipitation press has been patented. The idea is simple and consists of rotating a square or round fram press slowly, thereby obtaining a more even distribution of active precipitant in the press

frame and doubling the capacity per unit of space occupied.

The authors have given a clear picture of the method in use at Modderfontein "B" and their work deserves appreciation and the thanks of the members of this Society.

THE APPLICATION OF FLOTATION TO THE ANTIMONIAL GOLD ORES OF THE MURCHISON RANGE.

By H. R. ADAM.

(*Printed in Journal, July, 1922.*)

DISCUSSION.

Mr. E. C. Homersham had read with interest the paper by Mr. Adam. The author had given the results of a series of indication tests which he had made. These were interesting, but in no way conclusive, as far as flotation was concerned.

Given that the ore to be concentrated by flotation was a clean sulphide ore, there was little doubt no insurmountable difficulty would be found in producing a mixed concentrate containing 95 per cent. of the gold and antimony contents.

There were two reasons probably why these particular flotation tests gave unsatisfactory results:—

- (1) It was doubtful if coal tar and creosote, even with fine grinding, would give a clean tailing; it would certainly give a float, but probably other reagents would have to be used to get a clean tailing.
- (2) Type of machine used in test. He understood from reading the paper that the author had used a test machine practically equivalent to what is known as minerals separation standard type, which was certainly used with success on a great many ores.

Personally if he were experimenting on an ore of this description the speaker would use in the first instance a minerals separation sub-aeration machine. Probably two machines in practice would be used, one machine taking off a bulky float and the other being used to reclean primary concentrate, the tailing from the recleaning machine running back to the primary machine.

Should the antimonial ore be tarnished by weathering flotation would not be so easy and would be more expensive in reagents.

On any given ore it is necessary, before advising on type of flotation plant, etc., to do a series of preliminary tests to decide:—

- (1) The most economical and efficient reagents to be used.
- (2) The fineness to which the ore should be crushed to give the best commercial results.
- (3) Careful time test should be made to decide on the size of flotation plant required.

With regard to Test No. 3, it is evident that if an ore requires 10 minutes' treatment, and another class of ore takes 20 minutes' treatment, all things being equal, the plant to treat ore requiring 20 minutes' treatment should have a cubic capacity double that of the first. Near the surface there is in many cases a fairly large tonnage of semi-oxidised ore.

The author had made one very interesting experiment with amalgamation and then flotation. Recently he had received a sample of semi-oxidised antimonial ore. He first amalgamated the ore, then treated the residue by flotation, and the tailing which was still fairly high in value, but had all the antimony taken out of it, gave a very good recovery by cyanide. Mr. Simpson, who conducted the test, will, with permission, give the actual result.

In Mr. Simpson's experiment if he had used flotation prior to amalgamation free gold would have been mixed with the antimonial concentrate, and this of course is just what he would wish to avoid.

The results Mr. Simpson obtained were exceedingly satisfactory as far as they went. The practical point to be decided is what tonnage of this ore is available. No information on this point can be given as there is none, since the sample was sent to the laboratory by a client merely for testing.

Mr. T. R. Simpson said he had pleasure in giving the results of the test Mr. Homer-sham had just mentioned.

The ore consisted of stibnite, a small quantity of pyrite, oxidised compounds of iron and antimony associated with a siliceous gangue. A considerable quantity of the gold was coarse.

The crude ore assayed:—

Antimony, 8.3%.

Gold, 123.9 dwts.

This, of course, was an exceptionally rich ore. The best results were obtained by the following procedure:—

- (1) Amalgamation of the crude ore.

(2) Flotation.

(3) Cyanidation of the tailings.

Amalgamation.

A recovery of 53.6% of the gold was obtained by amalgamating the crude ore ground to pass a 40 mesh I.M.M. screen.

Flotation.

The residues from amalgamation were re-ground in a pebble mill to 5% plus 100 mesh (I.M.M.), and treated with certain reagents in a minerals separation sub-aerator machine.

The results were as follows:—

PRODUCT	Weight %	Assay Au dwts.	Recovery on crude ore %	Recovery based on flotation feed assay %
Crude Ore ...	100	123.9	—	—
By amalgamation ...	—	—	53.6	—
Flotation Feed (calculated)	57.5	—	—	—
Concentrate ...	29.4	162.8	38.6	83.2
Tailing ...	70.6	13.7	7.8	16.6

Cyanidation of Tailing.

The flotation residue was agitated for 24 hours with a 0.2% solution of KCN having an alkalinity of 0.05% CaO. After this treatment the residue had a calculated value of 5.45 dwts. This residue was again agitated for another 24 hours with the same solution; the final residue assayed 3.2 dwts. This shows an extraction of 5.9% of the gold based on the value of the original ore.

The recoveries are summarised as follows:

Recoveries based on value of crude ore.

By amalgamation ... 53.6%

By flotation ... 38.6%

By cyanidation ... 5.9%

Total ... 98.1%

The treatment outlined above was lengthy, but it should be borne in mind that an ore of this description in its raw state was not amendable to cyanidation; concentration was therefore essential to economical treatment, in that a smaller weight of ore would be roasted than would otherwise be the case.

NOTES ON ORGANISATION FOR PROMOTING INDUSTRIAL EFFICIENCY.

By S. DE SMIDT.

(Printed in Journal, August, 1922.)

DISCUSSION.

Mr. F. Wartenweiler said he would like to make a few remarks on organisation. He was interested in the paper by Mr.

de Smidt, but he thought the word "efficiency" was a much-abused one. He believed it had become so unpopular in other parts of the world that they had looked for substitutes.

He thought that efficiency must be considered as synonymous with effectiveness, and, to obtain effectiveness, they must also consider the human factor. Unless this factor was considered, and it was scientific to recognise it, all the systems for efficiency which they might propose and organise would fall to the ground. He found the personal touch was rather lacking to-day, particularly with the large corporations.

He recently had an experience with regard to the application of efficiency in taking samples of a certain product—say, a slime residue; the men, having had efficiency drilled into them, were very careful in returning an "efficient" residue by selecting only those charge cycles which they thought had received efficient treatment. Consequently there was an unaccounted-for gold loss, and he thought in this instance it was better to say less about efficiency and to impress honesty more.

Perhaps the author of the paper was not aware that the head office technical staffs of the mining groups were organised for efficiency work. He thought these organisations worked very largely on the question of efficiency, although they did not always call it by that name, but called it costs, extraction, and economics, and by other homely terms.

He was pleased the author had mentioned an organisation of one of the groups, "The Ideas and Suggestions Committee," which considered any meritorious ideas and suggestions brought forward on the mines of the particular group and voted suitable rewards.

Personally, he failed to see why efficiency engineers should always attack the mines; he saw far greater scope for them in other industries and organisations, for instance, in municipal technical departments. One need only observe the inefficient street sprinklers that were in use in this City, which washed rills and holes into the road surface after it had been carefully laid down.

The meeting then terminated.

BOOK REVIEW.

THE OIL ENCYCLOPAEDIA, by Marcel Mitzakis. Chapman & Hall, Ltd., 11, Henrietta St., W.C.2, London, 551 pp. Price 21/- nett.

This book had its inception in a series of articles contributed to the "Petroleum World." The present volume has been greatly revised and rewritten. Owing to the incomplete data published on this subject considerable difficulties were encountered by the author in compiling the present work. Government reports were largely drawn upon.

This encyclopaedia covers all the known oil fields of the globe, and its instructive explanation of technical terms used in the art of oil finding, winning, and transport is a particularly useful feature. It starts with the word "abandonment," and explains the causes of the loss of oil wells and the experience in different parts of the world with abandoned wells. Several pages are devoted to the subject of "drilling." Under "Galicia" the history and description of this important oil district is given. A page is devoted to "Viscosity" and its methods of determination. Altogether the author has succeeded in collecting a wonderful lot of information which must be useful and of interest to the student, and particularly to the oil specialist and investor.

A bibliography of related books and official publications is appended. F.W.

Notes and Contributions.

A NEW METHOD OF CLEANING VACUUM FILTER LEAVES.

By T. B. STEVENS.

It is with some diffidence that I call the method of cleaning vacuum filter leaves, which I am about to describe, a new method, for it has been in use for several years, although I have not yet seen any description of it published.

The usual method of removing the lime incrustation from filter leaves is by circulating weak hydrochloric acid through them. This is satisfactory as regards renewing the porosity of the cloths, but it is costly for acid, especially where the acid has to be transported any distance; it has also other minor disadvantages. At Kalgoorlie we

have found that with certain modifications the hydrochloric acid can be replaced with the less expensive sulphuric acid. The leaves are first removed from the filter vat and freed from adhering slime and any loose incrustations by washing and brushing in the usual manner. If the cloth is very hard a wire brush such as is used by fettlers for cleaning castings is a good tool to use, as it opens up the cloth. The leaves are next immersed in a bath containing a 2% solution of sulphuric acid, and allowed to soak in it for one hour. By this treatment all of the lime salts are converted into calcium sulphate and a considerable portion of it will be loosened and will fall away from the cloth. Some, however, will still remain in the fibre of the cloth, and this is removed by transferring the leaves to another vat and circulating fresh water through them for two or three days. At 18° C., one part of $\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$ will dissolve in 386 parts of water (*Rand Metallurgical Practice*, Vol. 1), or in other words, one hundred gallons of water will dissolve nearly three pounds of calcium sulphate. The ease with which it will dissolve and the cloth softened has to be seen to be thoroughly appreciated.

The plant required for this treatment is of the most simple kind. The acid bath, if built of sound timber, three inches thick, will last for years, and one capable of accommodating six leaves is large enough for any sized filter, as the time of contact required is so short. The water vat may be of any size, either round or rectangular, and needs to be fitted with a manifold pipe on one side to which the leaves are attached. In some instances I have forced the water through the frames, and in others I have used a small plunger pump to suck it through them. The choice of method is dependent on local conditions. The water after being used for this purpose can be utilised either to make up the mill circuit, or for the disposal of residue. Where water is scarce the same water may be circulated through the leaves until at least half saturated with calcium sulphate. The solubility of calcium sulphate may also be increased by the addition of common salt.

The softening process has been found to be just as efficient as when hydrochloric acid is used, for when the leaves are returned to the filter they will have the same duty. The cost of cleaning is very much less, for at Kalgoorlie, whilst the theoretical cost of dissolving one pound of calcium oxide with

hydrochloric acid is fourteen pence, the same work can be done with sulphuric acid for four pence. The sulphuric acid treatment has also the advantage that no acid proof apparatus is required to circulate the acid. Another minor advantage is that the corrosion of the pipe-work frames is also not so great as when hydrochloric acid is used.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

DISTRIBUTION OF SULPHUR FORMS IN A COAL SEAM.—Bulletin No. 125, issued by the University of Illinois Engineering Experiment Station and entitled "The Distribution of the Forms of Sulphur in the Coal Bed," compiled by Messrs. H. F. Yancey and Thomas Fraser, both of the United States Bureau of Mines, is a report of investigational work carried out by the United States Bureau of Mines under the terms of a co-operative agreement with the Engineering Experiment Station of the University of Illinois and the Illinois State Geological Survey. Previous work published by the Engineering Experiment Station on the forms of sulphur in coal was conducted in the Chemical Laboratory of the University by Messrs. S. W. Parr and A. R. Powell, who developed successful methods for determining quantitatively the different chemical forms of sulphur in coal. Their methods have been applied in the present work to the study of the manner of occurrence and distribution of the different forms of sulphur in the coal bed.

The experimental work was undertaken as a part of the Bureau of Mines plan of research on coal preparation, and the application of the information secured to the problem of producing cleaner coal was the principal object. The solution of this problem must of necessity rest to a considerable extent upon an intelligent knowledge and appreciation of the physical and the chemical forms in which sulphur occurs in a particular coal, and how it is distributed through the coal in the bed. This information is essential in forming an opinion on the extent to which any given coal may be cleaned by washing, that is, by a gravity separation in water. The forms of sulphur occurring in the coal are (1) sulphur as iron pyrites, (2) sulphur in organic combination with the coal substance itself, and (3) sulphur in sulphates, present chiefly as calcium sulphate. Pyrites occur both in macroscopic and microscopic form. Thiessen has found microscopic globules of pyrites in coals from many different fields in the United States, and even in peat.

Pyritic sulphur is not uniformly distributed in coal. It is well known that the variation in total sulphur content of the different benches or horizontal sections of a coal bed may be quite marked. The distribution of

organic sulphur has been given but little attention, because it exists in an amicroscopic form and must be determined by chemical methods. Until recently no satisfactory methods have been available. Some earlier work done by the authors since the publication of Parr and Powell's methods, has led to the tentative conclusion that the organic sulphur content of a given coal varies but little, and that, at least, it is much more uniformly distributed than pyritic sulphur. The distribution of these forms of sulphur has an important bearing on the practice of coal washing, the dry separation of impurities from coal, and the use of coal for coking purposes.

The fact that organic sulphur has some bearing on the washability of a coal has long been recognised. Some coal-washing reports have contained an approximation of the figure for the percentage of this form of sulphur, in some cases with fine pyrites included under the caption "fixed sulphur," "combined sulphur," or "organic sulphur." Bolling, in a Paper on "Chemical Control of Coal Washers," describing methods used at the plant of the Nova Scotia Steel and Coal Company, gives the following figures determined by a sink-and-float test on a calcium-chloride solution of 1.35 specific gravity. Raw coal mixture, total sulphur 2.30 per cent.; by calcium chloride, total sulphur 1.10 per cent.; $(2.30 - 1.10) = 1.20$ per cent. pyritic sulphur, organic sulphur 1.10 per cent. While this method does not give the correct values for organic sulphur, it does give a figure which is of value in coal-washing work.

Carl Wendell, in an article on coal washing, states concerning combined sulphur that many times in examining a new washing apparatus he had been assured by the manufacturer that the machine would produce the desired results, this conclusion being based on two figures only; namely, total sulphur content of the coal and combined sulphur content. Wendell adds that he invariably found that this premature statement was wrong. His comment concerning the values of such hastily formulated opinions is fully warranted.

It is apparent that the possibility of cleaning coal of sulphur by processes of dry separation such as selective mining, hand picking at the face and in the tippie, or by mechanical devices, depends upon the way in which the sulphur, particularly the pyritic sulphur, is distributed throughout the coal. This factor also has an important bearing in connection with the coking of coal. Uniformity of sulphur content is a valuable attribute of a coal to be used for coking or for gas manufacture. Some of the Illinois mines nominally rated as low-sulphur coal mines cannot be depended upon to deliver day in and day out a product consistently low in sulphur.

All of the mechanical or gravitational methods of cleaning coal depend upon the removal of particles of impurities of high specific gravity from coal which is of lower specific gravity. In this connection it is important to know whether organic sulphur segregates as does pyritic sulphur. In case concentrations of organic sulphur were found to exist it would be desirable to associate such occurrences with other impurities or specific recognisable conditions.

The observations set forth in this Bulletin are based on data secured by:—(1) Visual examination and sampling of coal *in situ* at mines in three different beds in the Central District. Sectional bench samples were taken in a number of places in each of these mines and analysed for total sulphur, organic sulphur, pyritic sulphur, and fine disseminated sulphur. (2) Investigations incident to a survey of the coal pyrites resources of the Central District, made by the Bureau of Mines and the State Geological Surveys of Illinois, Indiana, Ohio, Tennessee, Missouri, and Iowa, in 1918. A large number of mines were visited by the field men sent out by these agencies. Notes were taken on the occurrence of sulphur, and estimates were made of the available tonnage of pyrites which could be recovered and cleaned for industrial uses. The records of this work present quite a fund of information on the distribution of coarse pyrites in the various coal beds throughout the district. (3) Visual examination of the coal faces in mines in connection with other investigational work and routine face sampling.

A study of the data in this Bulletin suggests to the authors the following summary of conclusions.

Extreme irregularity of distribution is characteristic of the pyritic sulphur of coal in the bed. (1) The irregularities in the distribution of total sulphur are due mainly to the concentration of pyritic sulphur in coarse bands or lenses. (2) Fine disseminated pyrites are much more uniformly distributed than are coarse pyrites. (3) Microscopic pyrites were found in all the samples examined. The amount of pyritic sulphur in this form cannot be accurately determined because it varies widely even in the same thin section.

As compared with the great changes in pyritic sulphur content the vertical distribution of organic sulphur at a given location in the same bed is quite uniform. The results of this investigation, however, would indicate that in high-sulphur coals the uniformity of organic-sulphur distribution is confined to limited areas in the bed, and that taking even a single mine as a whole the variations in organic sulphur content may be considerable, although it is much more uniformly distributed than is the pyritic sulphur. In low-sulphur coals the variations in the distribution of organic sulphur are less and the values obtained for organic sulphur approach uniformity, even for considerable areas.

There is little evidence of a definite relationship in the occurrence of organic and pyritic sulphur. High-pyritic sulphur in a bench or section of a bed is not indicative of a correspondingly high organic sulphur content. The proportion of the total sulphur that is in organic combination in various coals varies within wide limits. In low-sulphur coals organic sulphur makes up a greater percentage of the total sulphur content than it does in high-sulphur coals. Organic sulphur makes up a greater proportion of the total sulphur content of coal than is generally recognised. In the 104 face samples taken in three coal beds, organic sulphur exceeded pyritic in 49 samples.

In estimating the washability of a coal the organic sulphur content is an important consideration. The chief value in experimental coal

washing of the determination of this constituent lies in finding a value below which there can be no reduction of sulphur content by mechanical processes of separation, since the organic sulphur content of coal is not reduced by such methods. Furthermore, in all coals a part of the pyritic sulphur will remain in the washed coal, the amount depending upon the physical form in which the pyrites occur in that particular coal. Thus in every case the minimum sulphur content that may be obtained in the clean coal is well above the organic sulphur content. All of the pyritic sulphur occurring in the form of microscopic pyrites, and a large part of that included under the classification of "fine disseminated pyrites," is not removable and for practical purposes in connection with coal washing may be considered as "fixed" sulphur. The organic sulphur content of some coals is sufficiently high to limit seriously the extent to which these coals can be cleaned of sulphur by washing. In some coal beds partial concentration of the pyritic sulphur in the form of thick bands or lenses at certain definite horizons makes it possible to clean the coal by selective mining or hand-picking. — H. F. YANCEY and T. FRASER, *Iron and Coal Trades Review*, July 21, 1922, p. 83. (J.A.W.)

TAR DISTILLING*—Up to about ten years ago town's gas was made almost exclusively in horizontal retorts at high temperature, and the resulting tar was thick and heavy. With the introduction of the vertical retort, a radical change occurred in the tar. The specific gravity was lower, and the tar of a more oily nature. Glasgow vertical tar yields on distillation 40.44 per cent. of pitch and 56.60 per cent. of oils.

Considering the method of formation of these two tars, the author remarks that the tar vapours in horizontal retorts have to pass through the hot outer crust of the charge, and also to some extent along the hot walls of the retort. Break-down occurs, and the resulting tar is benzenoid in character. In vertical retorts the earlier portions of tar vapours are provided with a cool means of escape from the retort through zones at a lower temperature than that at which they are formed. The tar resulting from such conditions is paraffinoid in nature, and is known as low-temperature tar. The later portions of tar vapours, however, are subjected to some extent to the conditions that appertain in horizontal retorts and give benzenoid tar.

Vertical-retort tar may therefore be looked upon as a mixture of high- and low-temperature tars. This tar contains less benzol than horizontal tar, practically no naphthalene, a large percentage of compounds extractable with caustic soda, containing very little real phenol. The following figures show the difference in distillation range between the tar acids obtained from vertical and horizontal retort tars. The whole of the oils up to pitch were extracted with caustic soda, and the resulting tar acids dried and then distilled.

Horizontal-Retort Tar.

The crude dry tar acids obtained were equivalent to 9 per cent. by weight of crude

tar, and had the following distillation range, using a Wurtz flask:—

Drop	186 deg.
45.2 per cent. at	200 deg.
69 per cent. at	210 deg.
72.5 per cent. at	220 deg.
81 per cent. at	230 deg.
84.5 per cent. at	240 deg.
88 per cent. at	250 deg.

Vertical-Retort Tar.

The crude dry tar acids obtained were equivalent to 13.2 per cent. by weight of crude tar, and had the following distillation range, using a Wurtz flask:—

	Cent.
Drop	185 deg.
10 per cent. at	200 deg.
28 per cent. at	210 deg.
45 per cent. at	220 deg.
56 per cent. at	230 deg.
63 per cent. at	240 deg.
78 per cent. at	280 deg.
84 per cent. at	300 deg.
92 per cent. at	310 deg.

Distilled above 26 deg. C ruby red in colour. Gave highly viscous fluid when cooled.

Tar distilling may be roughly divided into two parts: (1) The primary or crude distillation, and (2) the refining operations. The crude distillation is carried out in closed stills, and consists in evaporating from the tar a certain amount of its volatile matter, condensing this volatile matter, and separating it into fractions, leaving behind in the still a residue of pitch of good merchantable quality. The number of fractions made from the volatile matter varies, but the following may be taken as fairly average practice: (1) Crude naphtha, containing benzene, toluene, xylene, carbolic acid, etc. (2) Light oil, containing mainly carbolic and cresylic acids with some naphthas, and in the case of horizontal tars, naphthalene. (3) Carbolic oil, containing tar acids, mainly cresols. (4) Light cresote oil. (5) Heavy cresote oil or anthracene oil, containing anthracene, etc.

All subsequent operations are classified as refining operations.

(1) *Carbolic Extraction.*—Fractions, 1, 2 and 3 are washed with a 10 per cent. solution of caustic soda, prepared from lime and sodium carbonate. The caustic soda removes tar acids as their sodium salts. Treatment of the sodium salts with carbon dioxide liberates the tar acids and regenerates sodium carbonate, which is re-causticised with lime, and used over again. The crude tar acids are purified by fractional distillation.

(2) *Treatment of Washed Oils.*—The washed oils from Fractions 1 and 2 are distilled. The distillate known as "once-run naphtha" passes to the benzol refinery for further treatment. The residue, along with Fraction 3, is cooled in open pans and deposits crystals of crude naphthalene. The clear is oil drained from the naphthalene, and goes along with Fractions 5 and 6 to creosote storage.

(3) *Treatment of "Once-run Naphtha."*—The "once-run naphtha" is first washed (in mechanical agitators) with a 25 per cent. solution of sulphuric acid, and allowed to settle. The mixture separates into two layers; the lower

* Abstract of Paper read at the Annual Meeting of the Society of Chemical Industry, at Glasgow.

layer containing pyridine bases, as their sulphates is run off. The pyridine is recovered by treatment with ammonia with formation of ammonium sulphate and liberation of the pyridine. The pyridine-free naphthas are then washed as follows, allowed to settle, and the lower layer run off in each case: (1) With about 5 per cent. R.O.V. 168 deg. Tw. to remove thiophen, etc. (2) With water (without agitation); with water (with agitation), to remove as much acid as possible. (3) With caustic soda or sodium carbonate to neutralise any acid remaining. (4) With water to remove any excess caustic soda. The washed naphthas are then subjected to fractional distillation by means of steam in stills with efficient fractionating columns, and analysers, to separate benzene, toluene, xylene, etc. The author described in detail the type of plant adopted, and its layout at the Provan Works of the Glasgow Corporation.—W. A. WALMSLEY, *The Iron and Coal Trades Review*, July 14, 1922, p. 43. (J.A.W.)

NOTES ON THE CHEMICAL CONTROL OF CATTLE DIPPING TANKS.—Although it may be true that under practical conditions the amount of oxidation in dipping tanks is not usually serious, yet serious oxidation is often experienced or discovered if chemical control is closely practised. As an example: A dipping tank liquor showing in November arsenic in the form of As_2O_3 , of a strength of 0.107% out of a total of 0.111%; the following March had been reduced to 0.035% As_2O_3 out of a total of 0.148%.

The method used for determination, at Cedara Station, is as follows: The dip fluid is first clarified by the aid of a few c.c. of either strong HCl or H_2SO_4 . For the estimation of the arsenious oxide the filtrate, after neutralising with sodium carbonate, is titrated with $N/10$ iodine, after adding an excess of sodium bicarbonate as usual. In estimating the total arsenic, another portion of the filtrate is treated with a large excess of either concentrated HCl or H_2SO_4 and a grain or two of KI, in order to reduce the arsenate present to the arsenite condition. The reduction is complete after warming the flask for a few minutes. The free iodine is got rid of by means of sodium thiosulphate and the arsenic is then estimated as before.

A number of commercial standard solutions used for testing the arsenical dip were checked and found to be correct within 5% over three to four months. After that deterioration was greater, and after a year the standard solutions were useless. It is apparent that a large error can creep in by the use of old solutions for testing.

The effect of the presence of salts in the water used in dips was investigated, and it was found that water containing an appreciable amount of saline matter, especially salts of iron and of the alkaline earths, may cause a serious precipitation of both the oxidised and the unoxidised arsenic in the dipping tank.

Salts of sodium or potassium have no deleterious effect.—C. WILLIAMS, *South African Journal of Science*, December, 1921, p. 147. (F.W.)

METALLURGY.

MINERAL SULPHIDES AND ELECTRICAL ENERGY.

—That mineral sulphides may function both as attackable and unattackable electrodes in electrolytic cells has been demonstrated by a number of investigators, the work of R. C. Wells, as discussed in Bulletin No. 240 of the U.S. Geological Survey, being especially noteworthy. To show further that they may be a source of electrical energy of commercial value, and to stimulate interest in the solution of the chemico-mechanical problem, is the object of this paper.

In the accompanying table are tabulated the heats of oxidation of carbon, sulphur, the commoner mineral sulphides, and their metals.

COMPARATIVE HEATS OF OXIDATION.

Substance.	Product.	Components.	Total.	Calories.		Electrical Equivalent. Kw-hr. = kg-cal.
				Per kg.	Per kg. 863.	
C	CO	C+O	29.3	2,440	2.83	
C	CO ₂	CO+O	73.9	6,160	7.14	
C	CO ₂	C+O ₂	103.2	8,600	9.55	
S	SO ₂	S+O ₂	77.6	2,420	2.80	
S	SO ₂	SO ₂ +O	71.7	2,240	2.60	
S	SO ₃	S+O ₃	141.1	4,410	5.10	
Fe	Fe(OH) ₂	Fe+O+H ₂ O	68.9	1,210	1.41	
FeS	FeSO ₄	FeS+O ₄	210.9	2,390	2.77	
FeS ₂	FeSO ₄ H ₂ SO ₄	FeS ₂ +O ₇ +H ₂ O	352.0	2,930	3.40	
Zn	Zn(OH) ₂	Zn+O+H ₂ O	83.5	1,273	1.48	
ZnS	ZnSO ₄	ZnS+O ₄	205.0	2,345	2.73	
PbS	PbO	Pb+O	30.8	244	0.28	
Pb	PbSO ₄	PbS+O ₄	195.0	815	0.945	

It will be noted that, weight for weight at maximum oxidation, sulphur has about half the value of carbon and that the electrical equivalent of this heat is 5.1 k.w.-hr. per kilogram of sulphur. Unfortunately, sulphur is such a poor conductor of electricity that its use as an attackable electrode does not seem encouraging. However, in combination with the common metals, lead, copper, and iron, it exists in the form of mineral sulphides which are fairly good conductors, and which, as will be shown, may function as readily oxidizable electrodes liberating energy chiefly as electricity. Among these mineral sulphides galena is by far the best conductor, but, owing to its high molecular weight, equal amounts develop only about one-fourth the energy value of pyrite, which in turn is equal to about one-third its weight of carbon.

Carbon burned under boilers can only yield about one-third of its energy in the form of electricity, so that it is patent that pyrite and carbon are theoretically equivalent for power purposes if the heat of oxidation of pyrite can be changed directly into electrical energy, and practically so if the cost of conversion is the same. The power-house cost of changing the heat energy of coal into electricity is well known. No data are available upon which to base the cost of operating electrolytic cells of the type and size that the problem demands. However, it is known that, for instance, in the cyanide process, where large tonnages of ore are treated, the cost for agitation, filtration, handling, and so on, which are the prime factors in electrolytic-cell operation, is almost unbelievably small, being at the Homestake mine about 7 cents per ton. Possibly the most important point to be considered is the fact that quantities of copper pyrite are daily oxidised in metallurgical plants, resulting for the most part only in the discharge of sulphurous gases into the air, which, owing to the possibility of producing damage, have been the cause of costly litigation.

The cost for oxidation by roasting could be credited to any system that would produce equivalent results in recovering the metal from the ore. With pyrite, then, it is not so much a question of comparing it with coal, for here we have a substance that is essentially a waste product, which is already mined and delivered, and which must be oxidised. In doing this there is an available credit on account of the saving in treatment. Normally in the Butte district, Montana, about 4,000 tons of copper pyrite is oxidised daily in the production of copper. The energy capable of being generated by this pyrite, if there were no loss, is equal to

$$\begin{aligned} & 3,490 \times 4,000 = 13,960,000 \text{ kw. hr.} \\ \text{or } & \frac{13,960,000}{24} = 580,000 \text{ continuous kw.} \end{aligned}$$

This is worth, if we compare it with the best coal-burning practice,

$$\begin{aligned} & 580,000 \times \$20 = \$11,600,000 \text{ per year,} \\ \text{or } & \frac{11,600,000}{365} = \$31,600 \text{ per day,} \\ \text{or } & \frac{31,600}{4,000} = \$7.90 \text{ per ton of copper} \\ & \hspace{15em} \text{pyrite.} \end{aligned}$$

The proportion of this value that would be realised would, of course, depend on cell

efficiency, which might be anywhere from 50 to 90 per cent.

POSSIBILITIES OF PYRITE AS A GENERATOR OF ELECTRICITY.

The design of the laboratory cell described cannot, of course, be adopted on a large scale, but it serves to demonstrate the fact that certain mineral sulphides can be oxidised, giving up their oxidising energy in the form of electricity. It is interesting to note that as a source of electricity, pyrite has about twice the value of zinc, weight for weight, that the quantity of electricity generated by the oxidation of the sulphur of the sulphide is much more than enough to precipitate its metal from the sulphate formed. It may be possible to oxidise the sulphide in one cell generating electricity; and in another cell to use this energy to electroplate the metal from the solution that resulted from the action in the first cell. The reason for this excess energy of oxidation over that needed for metal deposition is evident when it is realised that when oxidising pyrite (FeS_2) seven atoms of oxygen are required, whereas when reducing FeO to Fe only one atom of oxygen is taken away.

In this article it has been pointed out that certain mineral sulphides are a possible source of electrical energy. Pyrite seems to be particularly promising for the purpose, especially in view of the fact that it is now oxidised metallurgically in enormous quantities, the result of which is often wasted. A simple laboratory cell has been described which demonstrates the possibility of oxidising mineral sulphides, with the production of electrical energy. Perhaps the only new matter herein developed is the fact that mineral sulphides such as pyrite oxidise readily as an anode in an alkaline solution.—G. N. LIBBY, *Queenstand Government Mining Journal reprint from Engineering and Mining Journal Press*, July 15, 1922, p. 270. (J.A.W.)

CASEHARDENING BY CYANIDE.—In most classes of machinery there are parts which are subjected to either friction, shock, vibration or alternating stresses, or a combination of one or more of these. Ordinary mild steels are often unsuitable for these parts, and in many instances special high-priced material is used. There are numerous cases where this is unnecessary and where casehardened mild steel can be substituted with economy and efficiency. Briefly, casehardening consists in heating steel to a red heat in a non-oxidising atmosphere, in the presence of nitrogen and carbon. The nitrogen forms nitride of iron on the surface of the metal. This is unstable and decomposes, leaving the surface slightly porous and in a favourable condition to absorb carbon. There is thus formed carbide of iron, of which the casehardened skin largely consists. Where a case of not more than 1-32 in. is sufficient the parts may be conveniently and economically treated in molten sodium cyanide.

Cyanide is a white crystalline solid, and before use must be fused in a welded steel vessel of a size commensurate with the work to be undertaken. In this connection it should be noted that depreciation of the molten bath depends to some extent on the surface exposed to the atmosphere.

and that a deep narrow vessel is therefore more economical than a shallow vessel of large diameter. A suitable arrangement is one having a cylindrical welded steel vessel built into a brick setting, and fired with gas or oil, a suitable flue being provided to carry off the products of combustion.

When the bath is hot (say, over 800 deg. C.) a whitish fume is given off; this is not poisonous, but is very irritating to the skin, and a hood, open at the front, should be provided to carry this into the flue. This may be conveniently done by encasing the brickwork in a light steel shell which is carried up about 18 in. above the level of the cyanide-containing vessel; the top of this shell should have a hole fitted with a loose door through which heavy work can be lowered into the bath by means of block and tackle. The bath should be maintained at a temperature of from 810 deg. to 820 deg. C., this is hot enough to ensure rapid action. Higher temperatures are undesirable, as the depreciation of the steel vessel and the contained cyanide increase rapidly above this point.

One hour's treatment in molten cyanide will produce a casehardened skin about 1.100 in. in thickness; thus for a case 1.32 in. thick about three hours' treatment is required. Before immersion it is necessary to ensure that the articles are clean, dry and free from grease. For this purpose they may be washed for a few minutes in a bath consisting of a 5 per cent. solution of caustic soda in water, which has been raised to about 100 deg. C. After washing and before immersion in the cyanide bath it is essential that the articles be absolutely dry, as any water introduced into the cyanide causes frothing and spluttering, accompanied by the liberation of undesirable fumes and possibly loss of cyanide.

The immersion in the cyanide bath carburises the surface of the article, the subsequent treatment to which it is subjected will depend upon the quality of steel used and the purpose for which it is required. With mild steels it will be found that for many purposes it is sufficient to quench the article in water or oil while it is still at the temperature of the bath, or in any case not lower than 760 deg. C. This will give a hard resistant skin and is suitable for articles subjected to friction of all kinds. Articles so treated also stand up well under alternating stress. Another method in common use is to allow the carburised article to cool after withdrawal from the cyanide bath, then re-heat and quench; the temperature of quenching depends upon circumstances, but will probably be between 750 deg. and 900 deg. C. An advantage of this second method is that the core or metal inside the hardened skin is greatly improved in quality and toughness.—*Iron and Coal Trades Review*, July 21, 1922, p. 83. (J.A.W.)

MINING.

USE OF WIRE ROPE IN MINING OPERATIONS.*—Every user of wire rope is desirous of information that will enable him to determine whether the performance of any particular piece of rope

* Read before the American Institute of Mining and Metallurgical Engineers.

is satisfactory, and what conditions can be changed so as to increase the life of a rope. In most cases, it is far from easy to analyse the conditions of service and tell which is the greatest factor in determining the life of wire rope. If we start on a lay-out for an entirely new rope system, the task of making conditions satisfactory is simplified so far as the engineering details are concerned, but if the plant requiring wire rope is in operation, any change is likely to involve the expenditure of considerable money to replace and change over machinery that must be kept in constant operation, because a large monetary loss will result from a shutdown for even a few days. Often wire rope is kept operating under bad conditions as the user figures that it is cheaper and quicker to replace the rope than to make alterations in the hoisting machinery.

Regular-lay and Lang-lay forms of rope construction are both old, and both have their advocates. In Great Britain, Continental Europe and South Africa Lang-lay ropes constitute probably 80 per cent. of all used in mining. In the United States probably 90 per cent. of all ropes in use are of regular lay.

In a Lang-lay rope the wires in the strands and the strands in the rope are twisted in the same direction, giving the rope the characteristic appearance of being loosely twisted. Each individual wire appears on the surface of the strand for a longer distance than in the case of regular-lay rope. The worn spots on any wire are longer and such ropes appear to wear down farther than on a regular-lay rope. In a regular-lay rope the wires in the strands have a left-hand lay and the strands in the rope have a right-hand lay. The rope has the appearance of being more closely twisted or laid together. A worn regular-lay rope exposes more worn crowns per foot, but the worn spots are shorter.

If a Lang-lay rope is twisted in a direction to unlay the rope, the wires in the strands will untwist at about one-third the rate of the untwisting that occurs in the rope proper, assuming the same angle of lay for each. As a result of such untwisting, the outside wires of the strands are loosened, the amount depending on the amount of untwisting. Excessive stretching of a Lang-lay rope will produce about the same effect. The only opposition, therefore, to the continued untwisting of a Lang-lay rope is the stiffness of the individual wires to the torsional stress tending to untwist them. If a Lang-lay rope is twisted in a direction to shorten the lay of the rope, there is at the same time a shortening of the lay of the wires in the strands equal to about one-third of the shortening of rope lay. The result of this tighter twisting of the lay is to produce a double shortening of the rope; consequently, a powerful force is exerted against a shortening of the lay compared to a weak force opposing the untwisting action. For this reason, a swivel on a Lang-lay rope will often cause undue untwisting.

A force tending to untwist the regular-lay rope meets violent opposition because the wires in the strands tend to twist together by an amount equal to about one-third of the twisting of the rope, instead of untwisting as in the case of the Lang-lay. When the operation is reversed and a regular-lay rope is twisted

together, the wires in the strands tend to loosen up.

Inasmuch as most wire ropes in service tend to lengthen in lay, or stretch, it is obvious that it is the behaviour of Lang-lay and regular-lay ropes to the untwisting forces that is of most importance to rope users, as this is the action that occurs most frequently in actual service. The question, therefore, as to the adaptability of Lang-lay or regular-lay to any rope installation must be settled first on whether both ends of the rope are fast at all times. If they are, no harmful untwisting should occur that would deform the Lang-lay rope.

In the case of a shaft-rope, safety demands that splicing must never be used; but on mine haulages, splicing occurs frequently as worn sections are replaced by new ones. The lack of familiarity with the splicing of Lang-lay rope is one of the drawbacks to its more extensive use in the United States.

If a wire rope drags or rubs, as frequently occurs on an incline or haulage, then, in general, Lang-lay rope will give the best results. Also, in cases of severe overwinding on hoisting drums, the Lang-lay rope will frequently give the best service. One fact, however, must not be overlooked, foreign rope practice on Lang-lay rope application is to use the largest possible diameter of sheaves and drums, which, of course, has a decided effect on the whole problem.

The most common method of attaching a wire rope to a drum is to pass the end of the rope through a hole in the drum and either clamp it to the drum with U-bolts or clamps, or to pass the rope around the drum shaft inside the drum and fasten the end back on itself. At least three complete wraps are usually left on the drum when the cage is down to the lowest level of the lift. This method seldom gives trouble, but in a few cases there has been a slight creeping of the rope at the point where the rope goes through the drum as the load comes off with the cage resting on the bottom, causing wear at this point of the drum.

A better method, and one requiring less rope, would be to cast a zinc cone on the end of the rope after it has passed through the drum and have the drum recessed so as to receive the end of the rope thus treated; then no slippage can occur.

For fastening the end of the wire rope to the cage or skip, different methods are used. The ideal cage fastening should develop the full strength of the rope. Of what value is a high-grade rope, designed to give a safety factor of 6, if the end fastening or attachment will give only 66 per cent. efficiency?

The various methods of making end fastenings on cages and skips may be classified as follows: wire rope clamps for forming small loop with thimble, wire rope clips for forming small loop with thimble, eye spliced in rope for forming small loop with thimble, and socket attachment.

A new method, the result of research work on the part of rope manufacturers seeking a more reliable method of socketing, is simple, cheap, and possesses the highest efficiency known

to the rope art. It is rated at 100 per cent. The important points are to clean the wires thoroughly, to flux them with acid, and to use zinc, never babbitt or lead. Zinc adheres to the wires firmly; in fact, the wires are galvanised into a solid mass. This new method of socketing is used exclusively by mines in South Africa for all their rope attachments.

The safety factor in a wire rope will be reduced by the following causes: (1) External wear; (2) breakage of individual wires; (3) corrosion by acid water, alkali water, electrolysis; (4) loss of elasticity; (5) fatigue in metal composing wires from regular or reverse bending, load variations caused by shocks due to starting and stopping or overloading, torsion at or near attachments.—J. F. Howe, *South African Engineering* (American Institute of Mining and Metallurgical Engineers), April, 1922, p. 77. (A.K.)

MISCELLANEOUS.

PERMEATION OF OXYGEN BREATHING APPARATUS BY GASES AND VAPOURS.—Question as to the permeability by gases and vapours of rubberised fabrics used in the breathing bags of self-contained oxygen breathing apparatus was raised recently by the death of James S. Cunningham, a foreman miner of the Bureau of Mines, who was overcome while wearing an oxygen breathing apparatus of the half-hour type in a large tank containing about 6 inches of gasoline at Trinidad, Colo. Investigation of the cause of his death indicated that the rubberised fabric of the bag had perhaps been penetrated by the vapour of gasoline. Experiments were therefore made with similar breathing bags in gasoline-saturated air. These experiments showed that when the gasoline used was highly volatile, like "casing-head" gasoline, dangerous amounts of it penetrated the bag. The materials now in use on standard breathing apparatus are quite impermeable to the gases ordinarily encountered in mine-rescue work, but do not afford complete protection in high concentrations of gasoline and other organic vapours. For such purposes special impermeable fabrics seem desirable.

Former Experiments on Gases Permeating Rubber.—Nearly a century ago, Dr. J. K. Mitchell, of Philadelphia, described his experiments that demonstrated that rubber films are permeable to hydrogen sulphide, cyanogen, ammonia, carbon dioxide, nitrous oxide, arsine, olefiant gas, hydrogen, oxygen, carbon monoxide, and nitrogen. He roughly measured the rates of passage of these gases through a thin rubber film and determined the speeds to be in the above order, hydrogen sulphide being the fastest.

Soon afterwards Draper found that hydrogen sulphide at atmospheric pressure would pass through a rubber membrane into a hydrogen-oxygen-nitrogen mixture of 24½ atmospheres pressure, "as readily as if no such force were exerted against it." Other gases acted similarly, and Draper concluded that a gas on one

side of a permeable membrane acts as a vacuum toward another gas on the other side.

Thomas Graham observed that "gases penetrate most readily which are easily liquefied by pressure and which are also generally highly soluble in water or other liquids." Temperature has the following effects:—It makes the gases less soluble, the rubber softer, and it increases the speed of diffusion; over a moderate range, 4 to 60° C., the combined effect of increasing temperature was to make the rubber film more permeable to air.

Studies of gases separated by soap and water films by Exner proved that the passage of each gas was proportional to its absorption coefficient or solubility in the liquid and inversely proportional to the square root of the density. Wroblewski demonstrated soon afterwards that the same relations hold for rubber films, and stated the relations with the formula:

$$Q = \Omega D S \frac{U_1 - U_2}{x} \cdot t$$

in which Q is the quantity of gas passed through a rubber of area Ω , and thickness x , in time t ; D is the diffusion coefficient of the gas; S the saturation coefficient of the rubber for the gas; and U_1 and U_2 measure the absorbed gas at the two surfaces of the rubber. His conclusions are in part as follows: Henry's law for gases and liquids apply for rubber and gases; the absorption coefficient for vulcanised rubber is a linear function of the temperature, for carbon dioxide and nitrous oxide it decreases with increased temperature, but for hydrogen it increases; the absorption is a purely physical phenomenon and not chemical; gases diffuse in rubber in accordance with the law that states the conductivity of solids for heat; this diffusion is independent of the chemical nature of the gas and is dependent only on the physical properties of the gas.

Recent work at the Bureau of Standards has confirmed many of the foregoing conclusions, and in addition has shown that rubber decreases in permeability with the degree of cure and with ageing. The "impedance," or reciprocal of permeability of a given amount of rubber is greatly increased when the rubber is applied to cloth; in some cases it is nearly doubled. Most compounding materials are kept low when high impedance is wanted, though glue and paraffin are said to lower the permeability. The "specific permeability" was defined as the volume of gas permeating 1 square centimeter of rubber 1 centimeter thick with difference of partial pressure of 760 millimeters on the two sides in one minute; for hydrogen the specific permeability at 25° C. is 20.4×10^{-6} cubic centimeters per minute. Relative powers of permeation of different gases were found to be as follows:—

Power of permeation of different gases.

Nitrogen	0.16
Air22
Argon26
Oxygen45
Helium65

Hydrogen	1.00
Carbon dioxide	2.9
Ammonia	8.0
Methyl chloride	18.5
Water vapour (approximately)	50.
Ethyl chloride	200.0

Discussion of the Results of Tests.—The experiments show that the thin rubberised coating on the apparatus tested (the Salvus apparatus worn by Cunningham) is rather easily penetrated by both liquid gasoline and by such high concentrations of gasoline vapour as may be expected in a tank containing some volatile casing-head gasoline. Although ordinary motor gasoline does not seem to produce high enough concentrations of gas to penetrate in dangerous quantities, in a period of 10 to 15 minutes a bag free from defects, yet the need of a much more impermeable bag is certainly demonstrated.

Actual tests of breathing bags of other apparatus have not been made, but all thinly-coated bags are permeable, as the later tests of fabrics show. Thus all oxygen breathing apparatus with such bags must be used with caution in high concentrations of gasoline or benzene and other organic vapours. A life line should always be attached to the wearer so that he can be immediately drawn out of the dangerous atmosphere should he show signs of distress.—A. C. FIELDNER, S. H. KATZ, and S. P. KINSEY, U.S. Bureau of Mines Technical Paper, 272. (C.J.G.).

CHANGES OF ADDRESS.

- BROCK, H. T., *U/o* Dynamite Factory, North Rand; B.S.A. Explosives Co., Ltd., P.O. Box 1938, Johannesburg.
- COLQUHOUN, L., *U/o* Dynamite Factory, North Rand; B.S.A. Explosives Co., Ltd., P.O. Box 1938, Johannesburg.
- HENNESSY, E. C. C., *U/o* Johannesburg; 2, Kyverlog Street, Cardiff, Glamorgan, South Wales.
- MARRIOTT, H. F., *U/o* London Wall; "Heids," Morland Close, Hampstead Way, London, N.W.11.
- MEIN, W. W., *U/o* New York; 2080, Gough Street, San Francisco, Cal., U.S.A.
- PIROW, H., *U/o* Mines Department, Krugersdorp; Mines Department, Johannesburg.
- RAY, L. G., *U/o* Mines Department, Germiston; Mines Department, Krugersdorp.
- WILSON, J. C., *U/o* Warmbaths; Messina, Zoutpansberg, Transvaal.
- WORMALD, R., *U/o* Noordkaap; Montrose Mine, P.O. Ivy, Barberton.

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THE VALUE OF RAND GOLD MINING INVESTMENTS.

By W. A. CALDECOTT and G. J. V CLARENCE.

The majority of the members of this Society are engaged in the technical operation of gold mines, and in view of this fact some discussion of an important economic aspect of mining may be of interest and service to them. The aspect referred to is that of the actual value of a mining enterprise, as expressed in the intrinsic or true value of the shares, the holders of which are the owners of the property. In this connection the fact may be recalled that all expenditure of capital, technical knowledge, inventive skill, and organising capacity in mining and metallurgy have for their dominant object the enhancement of the value of the mine. This increase in value ultimately takes the form of larger dividends paid to shareholders.

The popular belief that mining of gold in general is a pursuit as uncertain as it is fascinating has tended to obscure the fact that gold mines vary from the highly speculative order, particularly in the early stages, to those well-developed and rich mines which can be fairly described as offering one of the best forms of sound and profitable investment,* as well as constituting an easily realisable security. The popular practice is the purchase of mining shares, frequently after attention has been

attracted thereto by a considerable rise in price, for proposed re-sale at an early date, at a higher figure; and the more solid feature of profit from a steady and long-continued receipt of adequate dividends is not universally appreciated. It is hence the main object of this paper to discuss the latter aspect, and to emphasise the fact that the Witwatersrand gold mining industry, in addition to conferring many other benefits upon South Africa, regularly distributes wealth among its citizens in the shape of dividends amounting to considerable sums yearly.

Gold mining differs from other industrial undertakings in that the product represents money, and hence is not adversely affected by any depression which may reduce the market demand and price of all other commodities, because low commodity prices tend both to reduce gold mining costs, and to enhance the purchasing power of the gold produced and of the dividends distributed. As an illustration of the stability of a really good gold mine, the record may be recalled of the Robinson Gold Mining Co., which is now almost worked out, but which for a third of a century, and in spite of local and world-wide upheavals, has steadily yielded dividends amounting in all to twelve million pounds. An early investor in this mine, of which the capital consisted of 550,000 £5 shares, would have received on an average about 13 per cent. per annum on the par value of the shares, whereas the purchaser at the same early date of the gilt-edged security British Consols would only have received $2\frac{3}{4}$ per cent. interest from 1889 to 1903 and $2\frac{1}{2}$ per cent. since on the nominal value, and would now hold a

* "The Transvaal Companies in South Africa have, in many cases, blocked out sufficient ore to insure the repayment of the original investment many times over. Where this has been done, and the investor, therefore, has positive assurance that there is a large amount of ore actually in the possession of his company, mining stock may become almost as safe as any other form of investment and far more profitable than most." "Putnam's Investment Handbook" (1919), page 203.

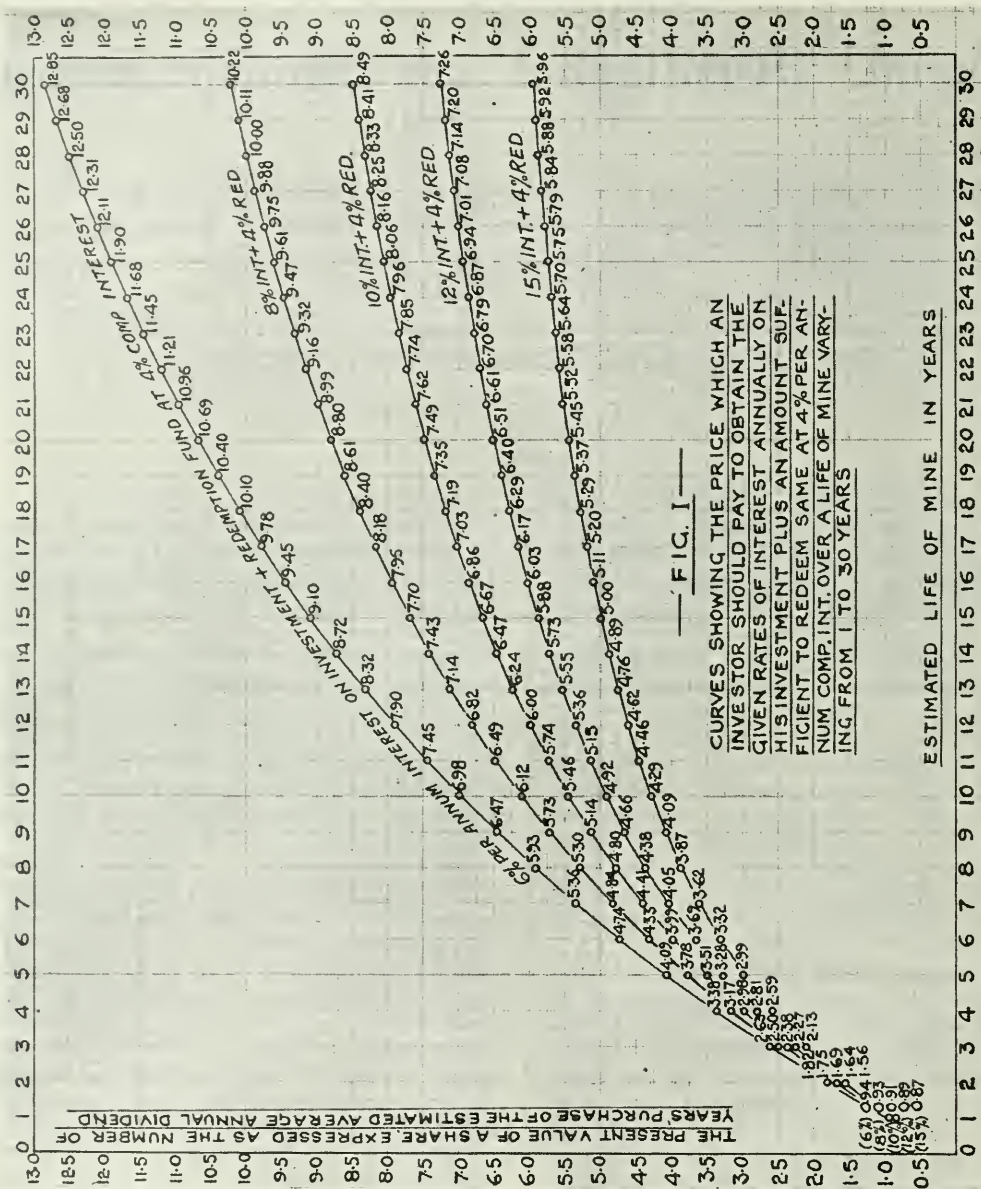


FIG. 1.

security standing at under 60 per cent. of par value.

At the same time mines such as the one instanced above are in the minority even on the Witwatersrand, and the exactitude with which mining share values can be calculated varies greatly. The degree of uncertainty is usually reflected in share prices, since investors require the rate of interest on the purchase price to be proportionate to the degree of risk involved. Further, owing

to changes in public feeling, the contraction or expansion of credit and currency, and the succession of the usual periods of the economic cycle—prosperity, decline, depression and improvement—the market prices of even the best mining shares usually vary considerably both above and below intrinsic values in the course of each year or two, even though the value of the ore reserves and of prospective dividends may be unchanged. The economic cycle may be compared to the motion of a pendulum whose

swing takes months or years, or to the alternate rise and fall of the tides. In other words, the market price of shares is the resultant of several causes besides intrinsic value, such as crowd psychology and fashion, the abundance or scarcity of money, political events, and temporary artificial factors such as advertisement. Hence, as in other forms of investment, material differences between market prices and intrinsic values occur, and are of prime importance to the investor. It is obvious that purchase during a period of depression and low market prices, such as occurred in 1921, would afford a far higher return than if made at the present time.* On the other hand, share prices may rise to a figure on which the dividend yield is no higher than the interest obtainable on other classes of stable investment, in which case retention of the shares is equivalent to accepting a permanent reduction of interest on capital invested during the whole life of the mine.

The method employed by mining engineers for mine valuation requires considerable experience and judgment and involves the calculation of the probable total tonnage and value of the payable ore in the mine, and hence its life with a given crushing capacity. Allowance is then made for future capital expenditure, Government taxation, percentage of waste sorting and gold recovery per ton, future working costs and other expenses, and finally the probable future monthly profits are deduced. The number of shares issued being known, the annual dividends can be calculated, and a share valuation thereby arrived at. However, the majority of investors are not mining engineers, and hence have largely to rely on published data.

The accompanying curves of Fig. I. are mainly applicable to mines whose dividend record, tonnage, and value of developed ore reserves, and probable life, are such as to

enable fairly close estimates to be made for the future by the investor who is not directly associated with mining operations.

For a well-proven mine with ore reserves developed some years ahead, the dividends of recent years serve as a criterion for the future, and such data is available in the periodical reports to shareholders of the mine itself, and in annual works of reference, such as Skinner's Mining Manual, and the annual reports of the Transvaal Chamber of Mines.

The reduced annual dividend in the last years of the life of a mine due to lesser ore tonnage milled may be offset finally by realisation of residual assets, such as sale of equipment second-hand, cash in hand, clean-up of plant, and other assets, for example, real estate. In fact, when a mine is nearly worked out the value of shares may mainly depend on the liquidation proceeds.

As a rough rule, about ten months' working profits or 83 per cent. of the gross yearly profits are distributed as dividends, the balance being absorbed in taxation, capital expenditure, renewals, etc., so that the monthly published returns give some indication of the next prospective dividend.

The prompt publication of accurate information by Rand mining companies greatly facilitates accuracy of valuation, and is probably unsurpassed by the practice in this regard of any other class of industrial enterprise.

The degree of accuracy with which the curves may be read is greater than either the estimated life can be calculated or the probable future dividends foretold. If, however, a greater degree of exactness is required than is afforded by the curves, recourse may be had to Inwood's, Hoskold's, or other standard tables. Rossiter's small work comprises tables for readily and correctly ascertaining the present value of shares, and also what dividends should be paid annually to justify the purchase or market price of shares. Calculations in this paper covering more than the thirty years shown in the curves have been directly based on the foregoing tables.

With regard to the curves themselves, as shown in Figure I., these have been plotted to show the more usual rates of interest, so as to enable the intrinsic and relative values of shares in mines with different lives and different annual dividends to be readily estimated and compared on the same basis.

* The following is the average percentage increase at 8th September, 1922, of the market prices of dividend-paying Rand shares above the average of their lowest prices in 1921:—Six mining corporations, 64.1 per cent.; ten Far East Rand mines, 55.0 per cent., thirteen other gold mines, 113.9 per cent. Hence the increased future return from the investment of a given sum at the earlier dates over and above that recently obtainable is expressed by the same percentage figures. For instance, if £1,000 or other sum invested now in an average Far East Rand mine yields 10 per cent. per annum the same amount invested at the earlier date would have yielded 15.5 per cent.

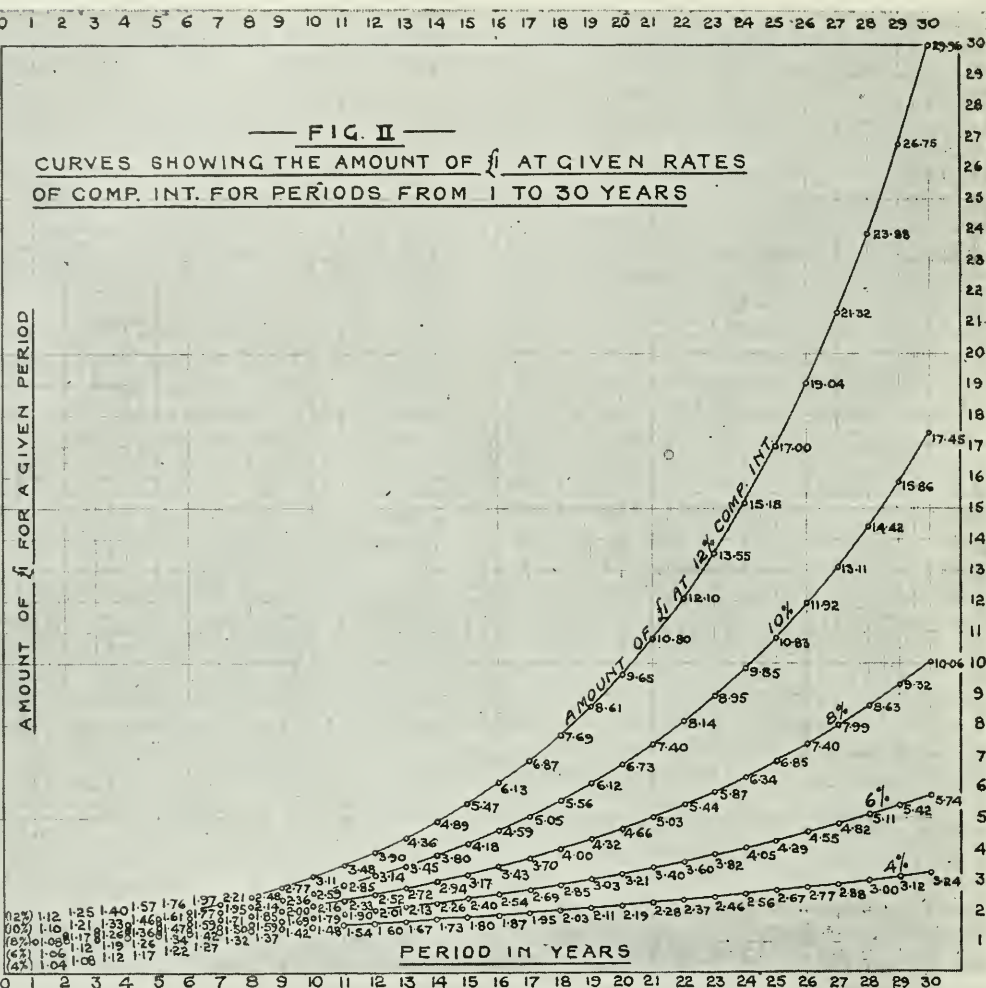


FIG. II.

If other intermediate rates of interest are required, such as 7 per cent., it is approximately correct to take the arithmetical mean of 6 per cent. and 8 per cent., or else the figure may be scaled off from the diagram. Should less than current rates of interest be anticipated in future years the present value of a long-lived mine may be calculated at a somewhat lower rate of income yield than for a mine with a short life. The abscissa in the diagram represents the estimated life of the mine in years, and the ordinate represents the present value of the share, expressed as the number of years' purchase of the estimated average annual dividend.

Five curves are shown illustrating the price which an investor should pay to obtain 6 per cent., 8 per cent. 10 per cent.,

12 per cent., or 15 per cent. interest per annum on his investment, plus a sum to be set aside annually at 4 per cent. compound interest, which will yield an amount equal to the purchase price of the share at the end of the life of the mine. Four per cent. redemption is taken as a conservative estimate of the return available by annual investment in a trustee security yielding low but safe interest over a long term of years, to provide for assured redemption of the capital originally invested. But anyone conversant with the highest class of mining stock on the Rand will have no difficulty in obtaining at least 8 per cent. on the redemption fund with consequent increase in the present value of his shares, as shown by comparing examples (1) and (18) given

later. In this connection it may be mentioned that small parcels of shares, say 5 or 10, termed "odd lots," can be readily bought or sold at slightly less favourable prices than the usual lots of 100 shares. This facilitates the regular re-investment of limited amounts of half-yearly or yearly dividends.

The working life of an individual resembles in some respects the productive life of a long-lived mine of say thirty years. After a period of capital expenditure in each case on development and equipment, production begins of a surplus of wealth above the cost of production. During this productive period the returns are partly income which may be expended, and partly redemption of capital; in the case of the mine of capital sunk in bringing it to the producing stage, and in the case of the individual of capital expended in up-bringing and training prior to working life, the redemption fund serving in practice in the latter case as a provision to ensure independent support for the individual in later years.*

The annual sum required for redemption equals the difference between the annual dividend and the required percentage income on the purchase price. In calculating the ultimate amount of the redemption fund it is assumed that the first instalment of this only becomes available a year after the purchase of the shares, and the last instalment of course synchronises with the receipt of the final dividend.

The share price as calculated from the curves is less than the true intrinsic value by an amount equal to the accrued dividend at any given date; for example, a price of 75/- as calculated on 30th September should be increased by 2/6d. if the mine is estimated to pay half-yearly dividends of 5/- each, as at 30th June and 31st December. In other words, one month's estimated

accrued dividend should be added to the curve price for each completed month since the end of the last dividend period.

It may be noted that the nominal value of a share, whether it be 1/- or £5, is not required to be known for the calculation of its present actual value, which depends only on the three factors:—(a) Annual dividend per share, (b) life of mine, and (c) rate of interest required, both as income and for the redemption fund. For instance, in example (1) below the share value will be 77.4/- whatever the nominal value of the share may be.

By the selection of two or more sound dividend-paying stocks, any temporary decrease of profits due to accident or other cause on a particular mine would not affect the whole of the income from this source.

The employment of the curves will best be understood by a few typical illustrations of their use:—

FIGURE I.

(1) To find the purchase price of a share, assume,

Life of mine = 22 years.

Annual dividend = 10/- per share.

Interest required = 10 per cent. per annum income on purchase price, plus an annual amount sufficient to redeem same at 4 per cent. compound interest over 22 years.

Then the purchase price = 10/- (annual dividend) multiplied by 7.74 (years' purchase factor for 22 years) = 77.4 = 77/5d.

Annual income = 10 per cent. on 77.4 = 7.74 = 7/9d.

Annual redemption = 10 - 7.74 = 2.26 = 2/3d.

2.26/- per annum invested at 4 per cent. compound interest for 22 years = $2.26 \times 34.2^* = 77.3\dagger = 77/4d.$ (purchase price of share).

(2) To find the annual dividend required to justify a given share price with a given life of mine and a given rate of interest, Assume,

Market price of share = 74/-.

Life of mine = 10 years.

Rate of interest required = 8 per cent. income + 4 per cent. redemption.

* "The ideal for the investor of moderate means is to secure a capital sum, the income from which will be sufficient to satisfy his needs without having to draw on the source of his income, and this should also be the ideal for the professional man without means, and others dependent upon an income arising from their own exertions, for, although it is possible to insure against a premature termination of such exertions, it is a good provision for old age or the time when they are past active work to have invested a capital sum, the income of which will provide for their needs during the rest of their lives." "The Stock Exchange Gazette," 2nd November, 1922.

* For the amount of £1 per annum for 22 years at 4 per cent. compound interest see Figure III.

† This is approximately correct; if greater accuracy is required reference should be made to tables referred to in the bibliography. The difference on 100 shares would amount only to 8/4d.

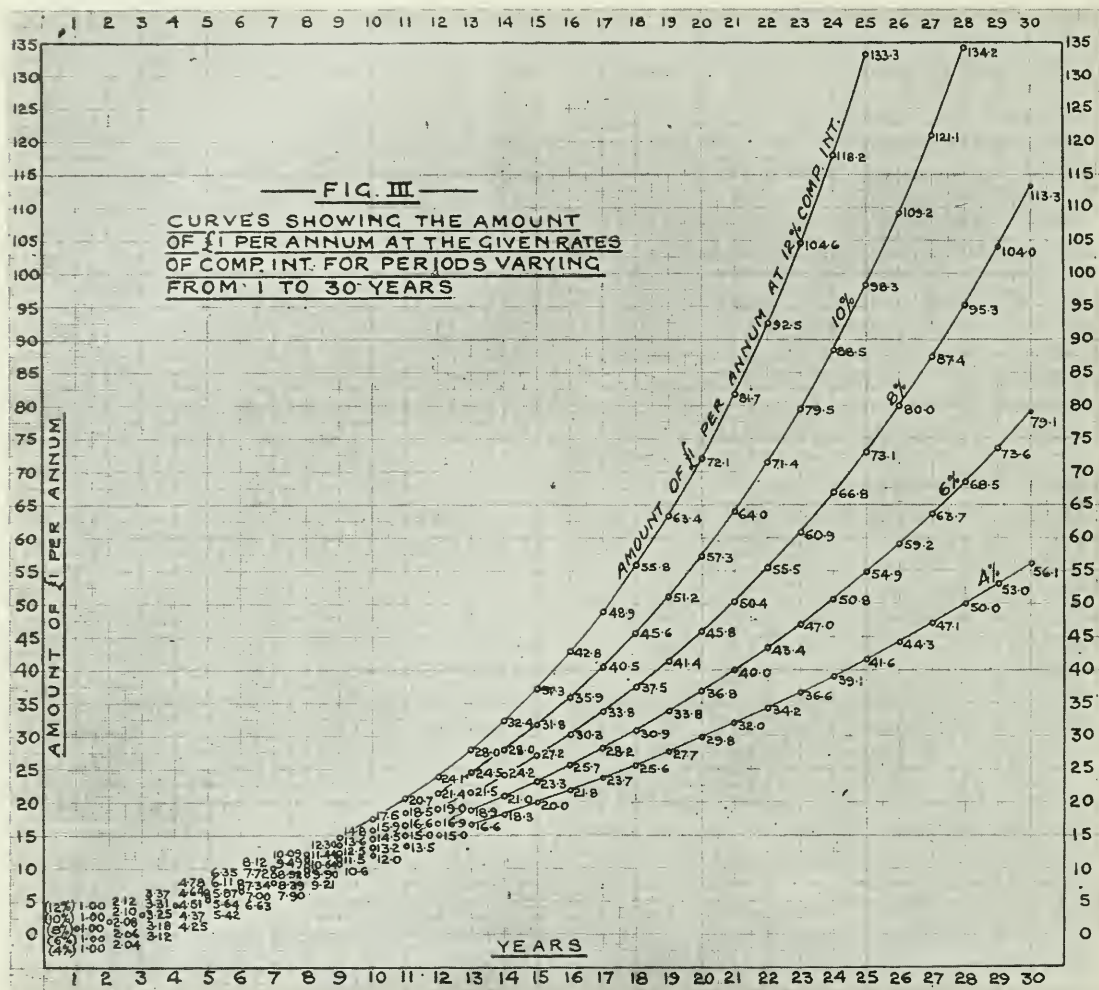


FIG. III.

Then the required annual dividend = 74/- (price of share), divided by 6.12 (years' purchase factor for 10 years' life on the 8 per cent. interest and 4 per cent. redemption curve) = 12/1d. (annual dividend required).

(3) To find the life of mine required to justify a given purchase price, with a given estimated annual dividend and a given rate of interest as income,

Assume,

Purchase price = 33/-

Annual dividend = 6/-

Rate of interest = 12 per cent. + 4 per cent. redemption.

Then the required life of mine is found as follows:—

33/- (purchase price) divided by 6/- (annual dividend) = 5.5 (years' purchase

factor for required life on the 12 per cent. interest and 4 per cent. redemption curve).

From inspection of the 12 per cent. income and 4 per cent. redemption curve it will be seen that 5.5 years' purchase requires a mine life of 12.8 years.

(4) The curves in Figure I. illustrate very clearly the increase in the present value of a share caused by a reduction of life, say, from thirty years to twenty years, through increasing the annual tonnage of ore milled. Assume an investor requires 10 per cent. interest with redemption at 4 per cent. in a mine paying an annual dividend of 5/- and having a life of thirty years.

Total dividend = 5/- × 30 = 150/-.

Present value per share = $5/- \times 8.49 = 42.45/-$.

Now assume that by increasing the crushing capacity, and disregarding both increase of capital expenditure and consequent reduced working costs per ton, the same total dividends of 150/- are paid in twenty years instead of in thirty years; this gives an annual dividend of 7.5/- over the shorter life.

Present value per share = $7.5 \times 7.49 = 56/2$, or an increase of 32 per cent. due to reducing the life by ten years.

Hence, in this case the percentage reduction in the life of a mine with fairly long life by increased rate of milling about equals the percentage increase in the present value of its shares. It is obvious, however, that this reduction of life cannot be carried on indefinitely, as the time factor for development as well as the capital cost of plant, supply of native labour available, and other factors must be considered in arriving at the most profitable life which is likewise practicable.

(5) At 8 per cent. income and redemption fund at 4 per cent. compound interest, the present value of the same annual dividends during the first ten years of a mine's life is equal to the present value of all subsequent dividends during the remainder of a century.

(6) Although the longer a mine's life the more difficult it is to determine its exact duration, yet fortunately any liability to error in this regard is largely offset by the small present value of dividends payable in remote years. On the basis of 10 per cent. income yield, plus redemption fund invested at 4 per cent., the present value of the dividends between the 14th and 20th years, or between the 20th and 30th years, or between the 30th and 54th years of a mine's life, is in each case only equal to one year's dividend in hand.

(7) With a long-lived mine it will be noted how slowly the present value of the shares decreases in the earlier years of its life. For instance, on the basis of 8 per cent. income yield, plus redemption fund at 4 per cent., it will be seen from the curve that with a life of 25 years, six years' dividends are obtained before the present value of the shares is reduced by the amount of one year's dividend, and the same applies to a mine with 20 years' life yielding 10 per cent. income, plus redemption fund at 4 per cent.

(8) With short-lived mines the curves show that a relatively small difference in the purchase price materially affects the income yield; for instance, with eight years' life a difference in price equivalent to only six months' dividend determines whether the investment will yield 10 per cent. or 8 per cent. income, since the difference between the factors is only 0.5.

(9) It will be observed from the curves in Figure I. that if four years' purchase of dividend is paid for a share, and redemption at 4 per cent. required, a mine life of about five years is needed for 6 per cent. income, five and a half years for 8 per cent. income, six years for 10 per cent. income, seven years for 12 per cent. income, and nine years for 15 per cent. income.

(10) If only 8 per cent. income is required, a mine with sixteen years' life is as valuable as a mine with twenty-four years' life when 10 per cent. income is required, the annual dividend and rate of investment of the sinking fund being the same in each case.

FIGURE II. AND FIGURE III.

Whilst the curves in Figure I. show how much the investor should pay for mining shares, Figures II. and III. show what return he should ultimately get for his investment, provided he re-invests his dividends at once when he receives them, and illustrate very clearly the truth of the old maxim that "time is money," and that the command of much money over a short period of time is equivalent to the command of much less money but more time. The curves show the amount of £1 and of £1 per annum at given rates per cent. For £1 we may substitute one shilling or one dollar or any other monetary unit.

The following are illustrations of the use of these curves:—

(11) To find the amount of £1 at 4 per cent. compound interest for 21 years.

From inspection of the 4 per cent. curve in Figure II. at 21 years £2.28 may be read off as the amount of £1 at 4 per cent. compound interest.

(12) To find the amount of £1 per annum at 8 per cent. compound interest for 14 years.

From inspection of the 8 per cent. curve in Figure III. at 14 years £24.21 may be read off as the amount of £1 per annum at 8 per cent. compound interest.

(13) Assuming that 8 per cent. interest on an average is obtainable from the best

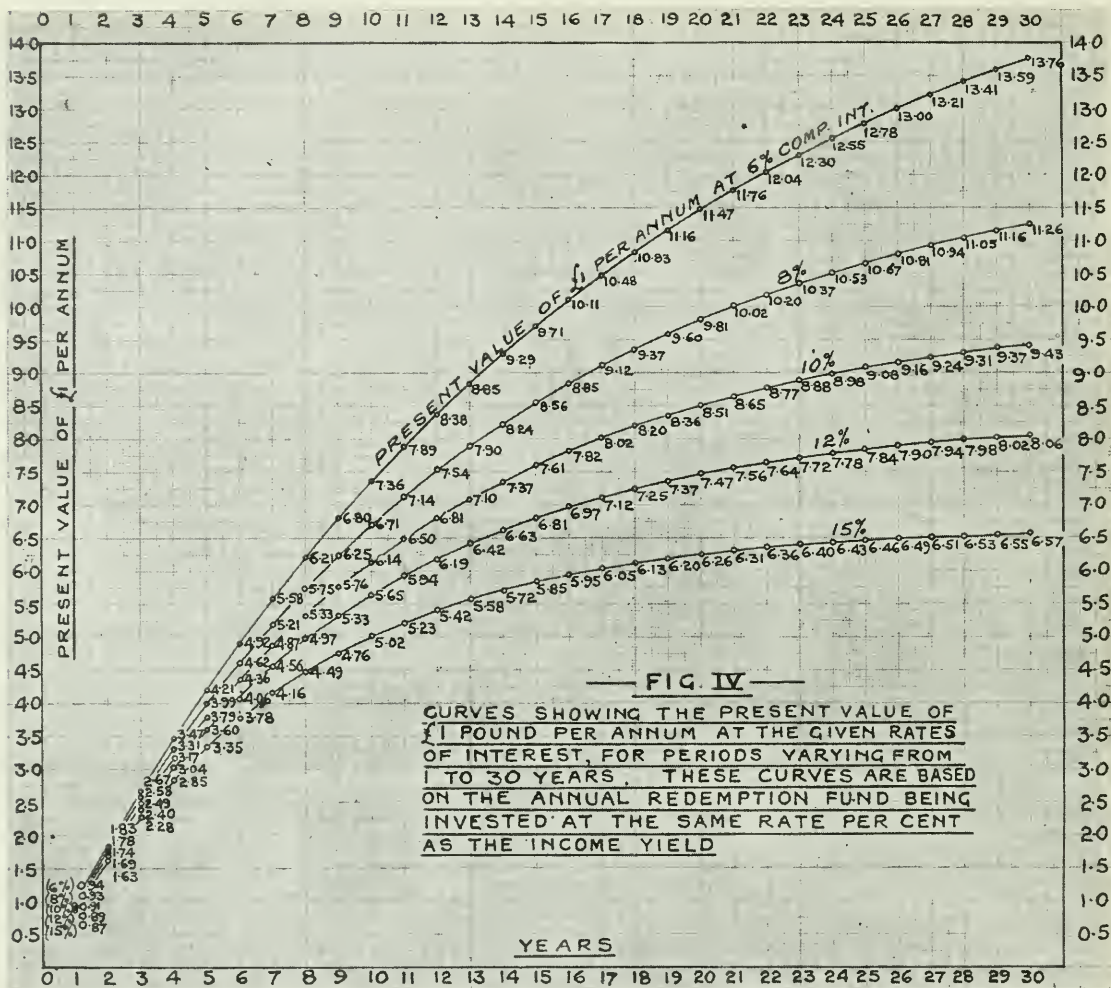


FIG. IV.

class of mining investment, and as £1 at this rate amounts, as shown by the curves in Figure II. to £4.66 in 20 years, £500 will hence yield £2,330 in the same period if the dividends are regularly invested at 8 per cent. In actual practice it may sometimes prove judicious to retain dividends for a time after receipt until the market price of shares has fallen, and under these conditions even more than 8 per cent. on the purchase price may be obtainable with corresponding increase of the eventual amount, which would reach £3,365 in 20 years from an original investment of £500 at 10 per cent. compound interest.

(14) The curves in Figure III. differ from those in Figure II. in that a yearly sum

is assumed to be invested at compound interest instead of a single initial capital sum. Assume that £50 per annum is available for investment at 8 per cent. compound interest as a provision for the future, or other purpose for which an endowment is required. After 25 years this would amount to £3,655, and even at the lower rate of 6 per cent. to £2,743. It will be observed from the steepness of the curves how rapidly the amount increases after the first ten years, thus again illustrating the importance of the time factor and the desirability of formulating and pursuing a definite financial policy before too late in the working life of the individual.

(15) A convenient approximate rule deducible from Figure II. for arriving at the amount of an investment after a period of years may be termed "the rule of 72." The rule is that the number of years required for any capital sum invested at compound interest to double itself equals the quotient of 72 divided by the given rate of interest. For instance, £1,000 invested in shares paying 9 per cent. income on the purchase price would, if the dividends are similarly invested, amount to about £2,000 in eight (i.e., $\frac{72}{9}$) years. In like manner the amount would be £4,000 in sixteen years, and £8,000 in twenty-four years. By reference to Figure II. it will be seen that the arithmetical mean of the 10 per cent. and 8 per cent. curves in the eight years vertical column is £1.995, or practically £2 as the amount of £1 in eight years at 9 per cent.

(16) A variant of the same principle is that any yearly sum invested at a given rate of compound interest will approximately amount to sufficient to yield the same sum in perpetuity at the same rate of interest in the number of years expressed by the quotient of 72 divided by the given rate of interest. For instance, if £100 yearly be invested at 8 per cent. compound interest it will amount in nine (i.e., $\frac{72}{8}$) years to £1.250, which at 8 per cent. yields £100 per annum. As will be seen by reference to Figure III. £1 per annum at 8 per cent. compound interest amounts to £12.50 in nine years, and hence £100 would amount to £1,250 under the same conditions.

A more exact form of the rule is that the number of years required as above equals

$$69.3$$

$$\frac{\text{Per cent. Comp. Int.}}{69.3} + 0.35^*$$

A similar rule employs 42 as the approximate factor to determine the number of years required for a capital sum to increase by 50 per cent. and the factor 24 for a 25 per cent. increase.

The foregoing factors may be used as a ready means, in the absence of curves or tables, of approximately determining the amount at the end of a stated period of a given sum, invested at any required rate of compound interest. For instance, what would be the amount of £100 in 23 years at 6 per cent. compound interest?

Original sum, £100.

Amount in 12 (i.e. $\frac{72}{6}$) years, £200 (100 per cent. increase).

Amount in 7 (i.e. $\frac{42}{6}$) more years, £300 (50 per cent. increase).

Amount in 4 (i.e. $\frac{24}{6}$) more years, £375 (25 per cent. increase).

The exact amount in 23 years is £381 19s. 6d., so that the error involved by the use of the above factors is under 2 per cent.

(17) The importance of the time factor in securing an endowment for future years may be further illustrated from Figure II., in that at 8 per cent. compound interest £1 invested now will amount to £10 in 30 years' time, whereas it would require £5 to yield the same amount at that time if investment be delayed for 21 years from now.

FIGURE IV.

These curves are of great importance to an investor in Rand mining shares who is able with the minimum of risk to invest his redemption fund at from 6 per cent. to 15 per cent. compound interest instead of 4 per cent. shown in Figure I. by purchasing shares of the best class with the excess of the dividends received over the required rate of income. The results attained by re-investment of the total dividends have already been illustrated by examples (13) and (15) of Figure II.

The effect of investing the redemption fund only as above will be readily understood from the following example:—

(18) To find the present value of a share, Assume,

Life of mine = 22 years.

Annual dividend = 10/- per share.

Interest required = 10 per cent. per annum income on present value, plus an amount sufficient to redeem same at 10 per cent. compound interest over 22 years.

Present value = 10/- (annual dividend) multiplied by 8.77 (present value factor for 22 years).

= 87.7/- (compare with 77.4/- in example (1)).

Annual income = 10 per cent. on 87.7 = 8.77/-.

Annual redemption = 10 - 8.77 = 1.23/-.
1.23/- per annum invested at 10 per cent. compound interest for 22 years =

$1.23 \times 71.4^* = 87.8/-$.

* See Schooling's "Value for Money," p. 136.

* See Figure III

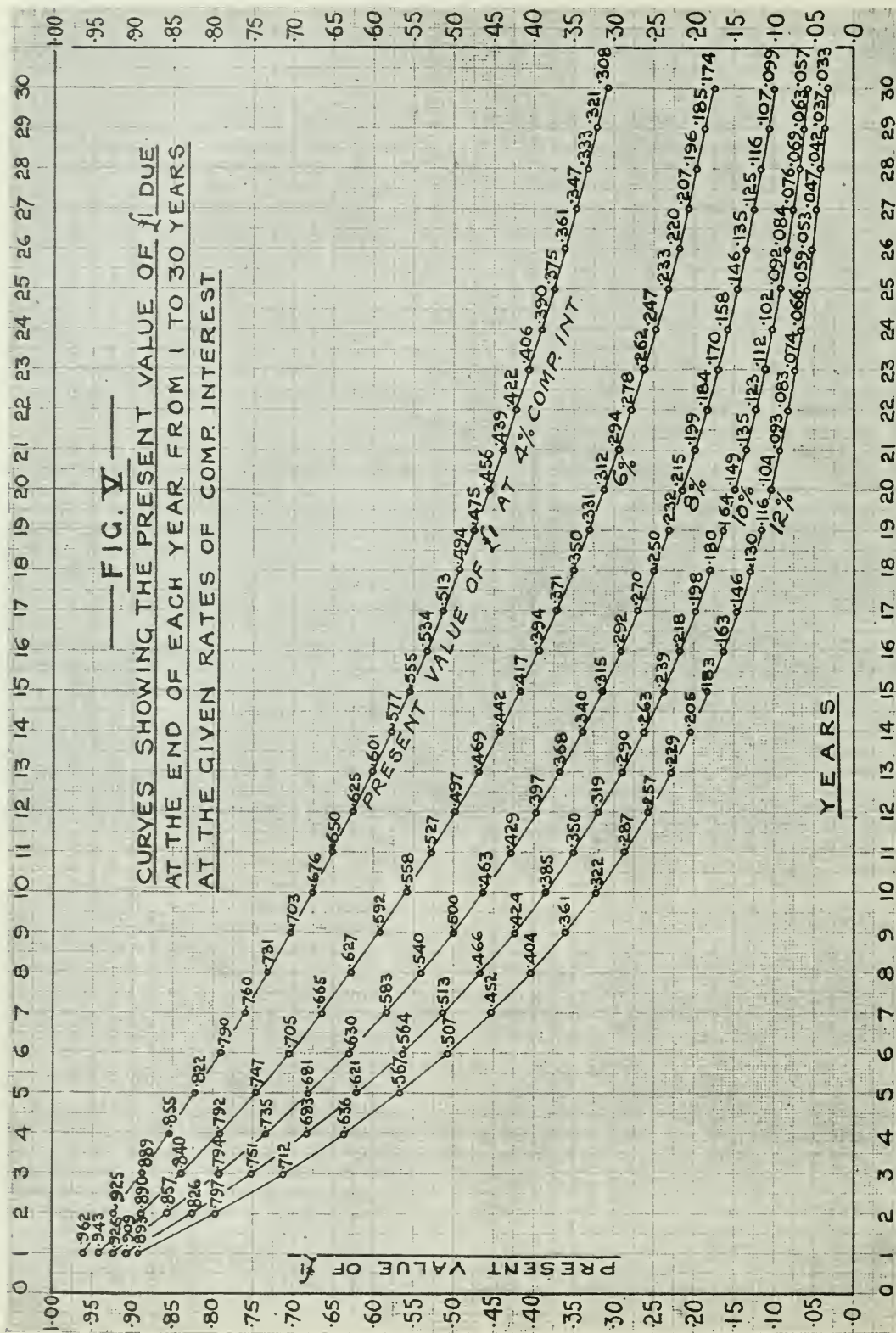


FIG. V.

Hence the present value as shown by the foregoing example, with the higher rate on the redemption fund, is greater by 13.5 per cent. than the value shown in Example (1); such increase is more marked with long than with short-lived mines.

As dividends are usually paid half-yearly and not yearly the present value of shares as given in Figures I. and IV. are too low by $1\frac{1}{2}$ per cent., 2 per cent., and $2\frac{1}{2}$ per cent. respectively, according to whether 6 per cent., 8 per cent., or 10 per cent. curves have been used. This, however, is generally regarded as an additional safety factor and is not reckoned in the estimate.

FIGURE V.

In the case of mining investments the curves in Figure V. are used in conjunction with those of Figure IV. for finding the value of debentures which, in addition to a fixed annual rate of interest, carry a guarantee of redemption, either at a fixed date or more usually over a period of years. If the debentures are redeemable over a period of years by equal annual drawings then it is customary to assume that the amount invested will be redeemed at the end of one half of the period remaining before the drawings are completed.

In valuing a debenture consideration must be given chiefly to the security of the investment, the interest yield of the debenture itself, and the price and date at which the debenture is to be redeemed.

The following example is taken as a fairly typical case:—

(19) Assume the rate of interest required is 8 per cent., the interest yield on £100 debenture is 6 per cent., and redemption is to be made at par by sixteen equal annual drawings, or on an average at the end of 8 years.

The present value of £100 at 8 per cent. compound interest due at the end of 8 years is from Figure V.,

$$£100 \times .540 = £54.$$

To this must be added the present value of £6 per annum at 8 per cent. for 8 years, which is taken from Figure IV. to be £6 \times 5.75 = £34.5.

Therefore the total present value of £100 debenture = £54 + £34.5. = £88.5, which is the price which an investor should pay to conform with the above conditions.

In conclusion, though the foregoing discussion is necessarily of a mathematical nature, the authors trust that the main principles and deductions have been set forth with sufficient clearness to be of some value to members. Reference to the works cited in the bibliography will afford further and more detailed information regarding both the subject matter of this paper, and also the considerations affecting investment in general, instruction on which is not included in ordinary scholastic training, and for which somewhat heavy fees are liable to be charged in the school of experience. The dominant aspect of the time factor in such matters has already been emphasised; to use a metallurgical simile, the dream of the old alchemists of transmuting silver into gold may be realised, but the reaction requires the passage of years to ensure completion.

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ALCOHOL FUEL MIXTURES.

By H. R. ADAM.

Although so much information has been published recently on alcohol fuels, particularly from the engineering standpoint, there exists some uncertainty regarding the composition of the various mixtures which have been used and as to the objects of adding other substances to the alcohol. It may therefore be of interest to examine the more typical mixtures with some reference to economic conditions in South Africa.

There is a tendency amongst those conducting efficiency and consumption tests to regard each new mixture, especially if it has been given some fanciful title, as a new and distinct substance. This tendency is passed on to the lay public by the daily papers by means of such phrases as "prickly pear spirit" or "vegetable petrol," as if the alcohol made from prickly pear were essentially different from that made from maize or molasses.

Every new mixture is rightly subjected to "road trials," but in considering the results of such tests it should be recognised that there are certain broad facts regarding alcohol used alone or mixed with such hydrocarbons as benzene or petrol, which have been well established by scientific tests carried out in many different parts of the world. It follows therefore that if the composition of the mixture is known the results to be expected can be gauged with reasonable accuracy and exaggerated reports of road performances modified accordingly.

The advantages and disadvantages of alcohol as a fuel are so well known that it is hardly necessary to re-state them, but in order to make the objects, of mixing other substances with the alcohol, clear the facts so far as they concern the question of mixtures may be briefly summarised.

CALORIFIC VALUE.

The round figure 11,000 B.Th.U. per lb. (nett) may be taken as an average value for industrial alcohol as used in South Africa for fuel purposes. The denaturants permitted ($\frac{1}{2}$ per cent. pyridine and $\frac{1}{2}$ per cent. suitable hydrocarbon oil) will if anything increase the calorific value rather than diminish it, as when a considerable proportion of wood spirit is required. If the calorific value were the only factor the con-

sumption ratio for alcohol to petrol (18,500 B.Th.U. nett) would be about five to three.

THERMAL EFFICIENCY.

With engines designed to give the best results from each fuel the thermal efficiency of alcohol is about 30 per cent. compared with 20 per cent. for petrol, so that the lower calorific value of alcohol is largely compensated for. This will not apply to the same extent when the alcohol is used at the much lower compression (about 70lbs.), which is usual in the engine designed for petrol. Nevertheless, according to the interim report of the Empire Fuels Committee of the Imperial Motor Transport Council, "at all speeds both with high and low compression the thermal efficiency obtained with alcohol was higher than that obtained with petrol or benzene . . . in spite of the higher thermal efficiency the fuel consumption of alcohol owing to its lower heat value was much greater." Until more definite figures are available we may take two-thirds of the mileage obtainable with petrol as reasonable for alcohol when used in petrol engines.

VAPOUR PRESSURE.

Alcohol has a much lower vapour pressure than petrol at the same temperature (e.g., for alcohol at 15° C. the vapour pressure in m.m. of mercury is 32.2, while that of a present-day petrol at this temperature would probably be between 100 and 150 m.m.). This means that except in warm weather there will not be sufficient vapourisation of alcohol to form an explosive mixture with air. To overcome this difficulty is one of the chief objects in making alcohol mixtures. Although a minor point with regard to actual fuel value it is of great practical importance, since reasonably easy starting from the cold is a necessity and the system of carrying a separate supply of petrol for starting purposes can never become popular. At temperatures above 20° C. there is usually no difficulty in starting with alcohol alone of about 95 per cent. strength, but for commercial purposes it is clearly necessary to provide a fuel which will start and give flexible running under all weather conditions.

CORROSION.

There is a vague idea prevalent that alcohol fuels are likely to cause corrosion

of the engine parts. As seen from the thermal efficiency figures alcohol actually gives more perfect combustion than petrol, and with good carburetting there is little to fear. Traces of acetic aldehyde and acetic acid may be present in the exhaust gases, but as both are highly volatile their corrosive effect must be negligible. Commercial alcohol always contains water, and rusting will result when alcohol fuels are allowed to lie unused in carburettors, tanks, etc. Alcohol, and more especially mixtures of alcohol and ether, have also a strong solvent action on varnishes, so that paint work and the varnished cork floats of carburettors are apt to suffer. These, however, are minor points, and on the whole the danger of any serious corrosion is slight.

It is obvious from the foregoing summary that the only practical objection to the alcohol itself as a fuel is its low vapour pressure, and even this objection would disappear during the hotter months in South Africa. A fuel is thus provided which will give about two-thirds the mileage of petrol. It is not sufficient, however, to sell the fuel at a price just sufficient to compensate for the larger consumption. Allowance has to be made for the natural prejudices of those accustomed to petrol; for fuel purposes the smell of petrol is preferred to that of alcohol.

MIXTURES OF ALCOHOL WITH BENZOL AND PETROL.

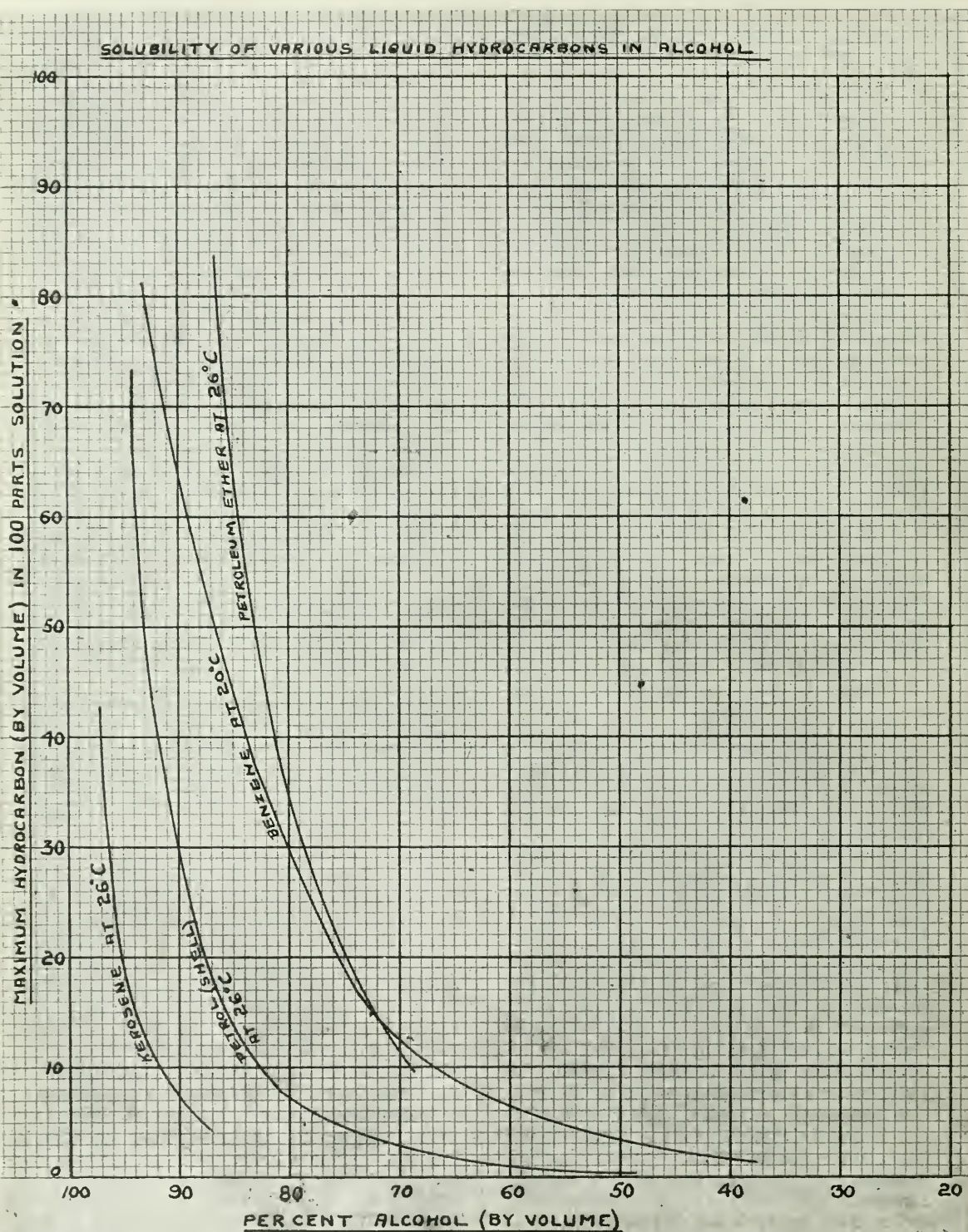
The mixtures most commonly used in France and Germany, and which have given satisfactory results in the extensive motor omnibus trials in London are mixtures of alcohol and benzol. The benzol is not pure benzene, which would be much too expensive, but a commercial product known as "90 per cent." benzol, which means that 90 per cent. must be capable of distillation below 100° C. It consists chiefly of benzene and its homologues toluene and the xylenes. The advantages of adding benzol are: (1) That with the increase in calorific value a corresponding improvement in consumption results. (2) The alcohol is denatured by the addition of benzol along with a small quantity of pyridine or an equivalent to make the solution sufficiently nauseous. (3) The smell of the benzol is familiar to the motorist and helps to remove prejudice. (4) Starting difficulties are lessened, though not entirely removed.

The data available are hardly sufficient to say whether the improvement in consumption is strictly proportional to the amount of

benzol added, but it is probably approximately so up to about 50 per cent. of benzol. According to the reports of the London General Omnibus trials a mixture of 70 per cent. alcohol and 30 per cent. benzol gave the best all-round results, this mixture giving a saving of 24.5 per cent. in British Thermal Units against petrol, in a straight test. The results with 90 per cent. of alcohol and 10 per cent. of benzol were not much inferior, which is the important thing for South Africa. Pagé¹ states that during the war the German motor service used a mixture of 50 per cent. alcohol and 50 per cent. benzol, which gave an average of 4.66 miles per litre as compared with 3.6 with petrol. With five parts alcohol to one part benzol 3.7 miles per litre were obtained. Considerable trouble in starting was experienced with these mixtures, so that a secondary tank containing petrol or ether had to be fitted. It is clear, therefore, that alcohol-benzol mixtures form an excellent fuel, and the best proportion of benzol will be decided chiefly by the relative cost of the latter. The improvement in mileage obtained with 50 per cent. benzol as compared with 10 per cent. does not seem sufficient to warrant so much being added where it is much more costly than the alcohol as in South Africa at present.

In place of benzol petrol may be used within limits, and the effect will be much the same. With regard to petrol additions a point of considerable importance arises which applies to a lesser extent with benzol, namely, the amount of water in the alcohol.

Recent work, particularly that of the Empire Fuels Committee, has shown that the presence of a considerable proportion of water in the alcohol is by no means disadvantageous. Apparently the decreased heat value is compensated for by smoother running and greater efficiency. In any case the enormous economic importance of this result is obvious since 85 per cent. alcohol may be practically as good as 95 per cent. The amount of water in the alcohol is, however, the chief limiting factor in the addition of petrol and benzol. The curves (Fig. 1) show how rapidly the solubilities of these hydrocarbons in alcohol fall with increasing water. In the figure only one petrol solubility curve is given since, as may be seen from the accompanying table, other brands give very similar results; a rather surprising result as there are considerable differences



- FIG. 1 -

TABLE I.
Solubilities of various hydrocarbons in different strengths of alcohol at 26° C.

Per cent. Alcohol by vol.	Maximum hydrocarbon forming homogeneous solution.							
	Petrol (Shell)	Petrol (Zenith)	Petrol (Pegasus)	Petrol (Triumph)	Kerosene	Benzene* (pure) at 20° C.	90% Benzol	Petroleum Ether
97.0	Completely miscible	Completely miscible	Completely miscible	Completely miscible	39	Completely miscible	Completely miscible	Completely miscible
95.0					19			
94.0	66	63		69		93% Alcohol 79		
92.5	45	41	44		11		90% Alcohol 82	
88.0	21	18	20	18	5	89% Alcohol 59	72	82
84.5	12					83% Alcohol 37		67
81.0	8	8	8					37
78.0	6							
75.0		4						20
72.0	3					14	25	
65.0	2			1				

* Calculated from figures given by Seidall "Solubilities of Inorganic and Organic Substances."

chemically. These are approximate figures only, as the method adopted (titration of the alcohol with the hydrocarbon to a permanent turbidity) does not give a very sharp end-point. In the case of the very light petroleum ether no turbidity results, but an immediate separation into two layers which is easier to detect if a little colouring matter, soluble only in the alcohol layer, is added.

The solubilities of petrol and benzol in alcohol also increase with the temperature, although the effect is not nearly so great as that of water in the alcohol. The curve (Fig. 2) illustrates the solubility of an American petrol in 89 per cent. alcohol at different temperatures. A disadvantage of benzol which applies in cold countries is the freezing out of the benzene at very low temperatures.

The addition of petrol or benzol to the alcohol will favourably affect the vapour pressure from the point of view of starting from the cold. Quantitative data are required, however, in order to say what degree of improvement results, especially in presence of different proportions of water. Young and his collaborators² have determined the boiling points of mixtures of the lower alcohols, benzene and water, but these researches deal with pure benzene, and for practical purposes definite information is required for mixtures of the commercial products. That the starting troubles are not entirely removed is evident from the statement by Pagé already referred to and also from the report of the London General Omnibus trials, which, however, puts the trouble down to an inferior supply of benzol. Dietrich³ also states that with alcohol-benzol

mixtures a preliminary heating of the carburettor is required. The practical importance of this point is emphasised by the favourable results obtained with alcohol containing a considerable proportion of water.

MIXTURES OF ALCOHOL WITH ETHER, ACETYLENE, ETC.

Coming to additions other than hydrocarbons we have various mixtures containing ether. Ethyl ether is commonly used, and is miscible in all proportions with alcohol. On account of its low boiling point (38°C.) about 10 per cent. is usually sufficient even for cold weather. Mixtures of alcohol and ether have been used in France as far back as 1908; thus a fuel known as "E.H.A." contained alcohol with 10 per cent. ether and up to 25 per cent. benzol. In South Africa an alcohol ether mixture (Natalite) containing about 40 per cent. of ether is well known. The vapour pressure of such a mixture would probably be much higher than is necessary for overcoming starting difficulties.

Since ether has a higher calorific value than alcohol (about 15,000 B.Th.U. per lb. against 11,000 for alcohol) the calorific value of a 40 per cent. solution would be about 12,500 B.Th.U., and a correspondingly greater mileage should be obtainable, although it would appear that the same result could be effected more cheaply by the addition of petrol or benzol. According to the Board of Trade Journal⁵ the large proportion of ether is considered a drawback for economic reasons. It should be pointed out, however, that in the manufacture of fuel it is not necessary to make ether free from alcohol, which should materially lessen the cost. Indeed it has been proposed in one English patent application to simply "etherise" a portion of the alcohol during distillation of the latter for fuel purposes.

Another method to improve starting which has been proposed, and which has been applied in South Africa in the fuel known as Penrol, is to saturate the alcohol with acetylene gas. According to Juritz⁶ the objection to this is that acetylene is practically insoluble in alcohol so that the mixture is unstable. In the fuel known as "Acetol" this objection is said to be overcome by the addition of acetone in which acetylene is considerably more soluble. The criticism of this proposal must be that acetone is nominally rather expensive. The importance of keeping the cost of alcohol fuels low cannot be too strongly emphasised.

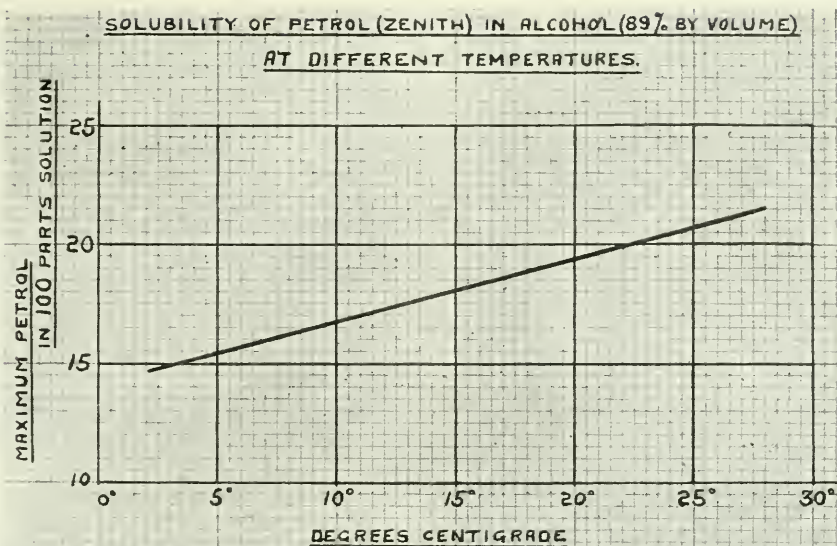
A method proposed and experimented on by the writer in collaboration with Mr. J. M. Thorburn is to dissolve a small proportion of a lower ether, viz., methyl ether, in the alcohol. It was found that at the atmospheric pressures prevailing in Johannesburg and at temperatures in the neighbourhood of 20°C. the solubility of this gas in alcohol was about 9 per cent. by weight (i.e., a saturated solution would contain about 9 per cent. of methyl ether). Solutions prepared by diluting a saturated solution with more alcohol so as to give proportions of from one to two per cent. methyl ether gave no trouble in starting. The curve (Fig. 3) is plotted from determinations of the vapour pressures of alcohol (93 per cent. by weight) containing different proportions of methyl ether. The curve is an approximate one since the proportions of methyl ether present in solution were not known with great accuracy; the actual figures obtained are shown in the accompanying table. The mark on the curve at 130 mms. of mercury represents the vapour pressure of an American petrol compared under the same conditions, and, according to this data, a little over two per cent. of methyl ether would be required to give an equivalent vapour pressure.

The addition of methyl ether will, of course, have no effect on the calorific value, so that the consumption of such a solution is practically identical with that of alcohol of the same strength. Benzol, petrol, or even kerosene can be added, and since the gas has a fairly high solubility in water it may be anticipated that alcohol of, say, 85 per cent. would simply require a little more of the gas. In fact this is just the acetylene idea except that the gas used, being much more soluble, a solution considerably removed from saturation can be kept indefinitely.

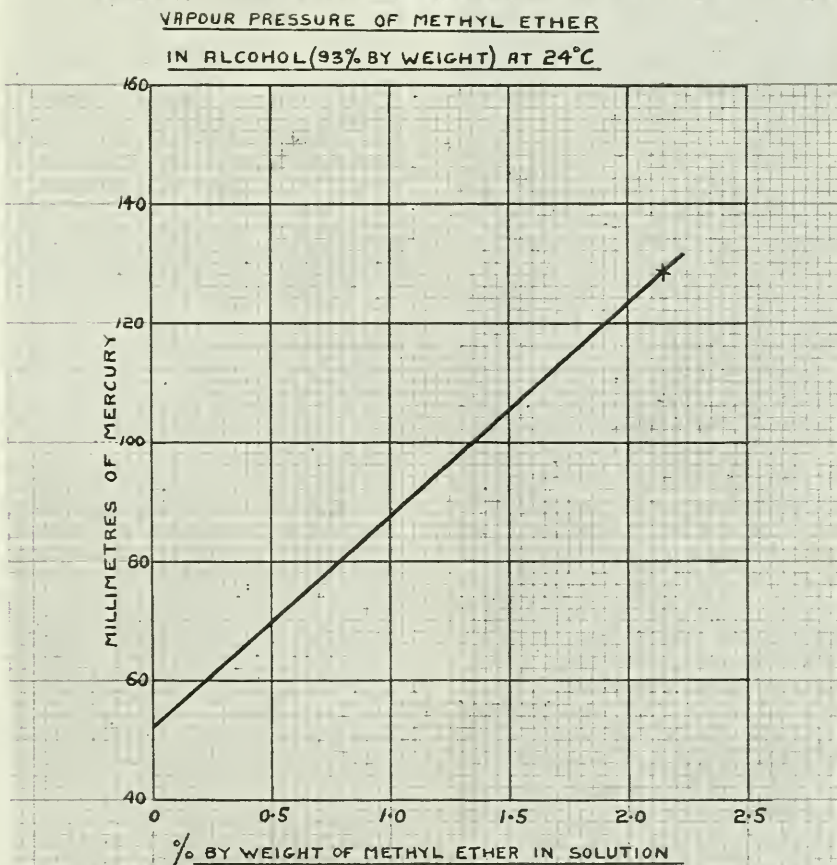
Methyl ether is made in a manner exactly analogous to ordinary ether, methyl alcohol instead of ethyl alcohol being used.

MIXTURES CONTAINING KEROSENE.

Recently in America a number of patent applications have been made for mixtures of alcohol and paraffin oil or kerosene. As will be seen from the kerosene curve (Fig. 1) its solubility in alcohol is very limited, but by the addition of a third liquid, such as ether, mixtures are made which are not only homogeneous but have increased volatility. At present this method cannot have much interest for South Africa, where kerosene is nearly as costly as petrol; the idea may,



- FIG. 2 -



- FIG. 3 -

TABLE II.

Vapour pressures of Methyl Ether in Alcohol (93% by weight) at 24° C.

Per cent. Methyl Ether.	Mms. Mercury.
Nil	52.0
0.5	66.1
1.0	88.6
1.5	101.2
<i>Petrol</i>	130.0

however, be useful if kerosene from the distillation of oil shale should be produced in the future. The objections to kerosene for motor vehicles are that, like alcohol, its vapour pressure is too low, and in addition its combustion is even less perfect than that of petrol. These objections should be largely overcome by such a method as that indicated. At the same time the information as to how such complex mixtures behave in running is as yet rather meagre.

CONCLUSION.

The questions involved in the cheap production of industrial alcohol are of infinitely greater importance than the particular mixture to be used. If commercial benzol is produced in South Africa at a reasonable cost we probably could not do better than follow the lead of France and Germany and use alcohol benzol mixtures. Alternatively petrol can be added, though this is simply a compromise and will add correspondingly to the cost. In any case the individual preference of consumers will no doubt settle the question and provide scope for several mixtures. The starting difficulty is probably not very serious in this country, and in addition to the methods already indicated it may be expected that if alcohol fuel is produced in large quantities some simple heating device for giving a preliminary warming to carburetters would soon be developed.

For agricultural work and heavy transport where facilities are provided for running on kerosene there does not seem any reason why alcohol alone of 85 to 90 per cent. strength should not be used, and this may be found the most economical in the long run.

It is not the purpose of this paper to discuss the big question of alcohol production. It may, however, be pointed out that in

South Africa the difficulties are chiefly those of organisation to make the best use of the abundant raw materials rather than those of actual technique, which has reached a high state of perfection.

REFERENCE LIST.

- ¹ Pagé: "Gasoline and Kerosene Carburetters," p. 29.
- ² Young and others: Trans. Chem. Soc., Vol. 81 (1902).
- ³ Simmonds: Alcohol; its Production, Properties, and Uses, p. 384.
- ⁴ Ibid.
- ⁵ British Board of Trade Journal, Sept., 1919.
- ⁶ Juritz. S.A. Jour. of Industries, Vol. III., Oct., 1920; Vol. IV., Dec., 1921.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

CORROSION OF METALS: Summary of Sixth Report to the Corrosion Research Committee of the Institute of Metals on the Nature of Corrosive Action, and the Function of Colloids in Corrosion, by Guy D. Bengough, M.A., D.Sc., and J. M. Stuart, M.A., presented at the Annual Meeting of the Institute of Metals, held in Swansea, on Wednesday, September 20th, 1922.—The difficulties encountered by the electrolytic theory are indicated, and it is shown that it gives a satisfactory account of the facts only under certain conditions, while many facts can only be explained by recognising the important part played by *colloids* in corrosion. A theory of the mechanism of colloid action is put forward, and some experimental results are reviewed in the light of this theory.

Corrosion is defined in its widest sense as the oxidation of a substance. It is then pointed out that such oxidation may be produced by chemical or electro-chemical means, and these two types of reaction are defined. Chemical reactions may occur when the reacting bodies are in contact, electro-chemical reactions when the reacting bodies are spatially separated. In the latter case the reacting substances must be capable of ionization, and a portion of the energy of the system appears as electrical energy. Two cases of corrosion are considered, both of which can be carried out chemically or electro-chemically. Pure electro-chemical action may in certain cases be relatively unimportant. Thus the cathode of a cell of high voltage may be more rapidly attacked than the anode, while an anode at a high voltage tending to force it into solution may be very little corroded, owing to scale formation.

Further facts which are difficult to explain on a purely electro-chemical theory are the following:—

(1) Certain depolarizers do not increase corrosion, but actually inhibit it.

(2) The conductivity of electrolytes is not directly connected with the amount of corrosion.

(3) Lambert's pure iron (probably the purest metal ever produced) was found to be readily attacked by sodium chloride solution and dilute acids.

(4) According to the electro-chemical theory, the presence of ions of the corroding metal should depress the corrosion of most of the common metals. There are, however, numerous exceptions, and in some cases the presence of such ions actually increases corrosion.

The order of corrodibility of metals in distilled water, certain salt solutions, and non-electrolytes is different from their order in the electro-chemical list, which suggests that there are factors interfering with the electro-chemical action. Such a factor is scale formation, and a main factor in determining the amount of corrosion by water and salt solutions is the nature and distribution of the products of corrosion. This may be far more important than any hypothetical distribution of cathodes and anodes in the metal.

The effects of strain and impurity in the metal are considered on the electro-chemical view to be of fundamental importance, and Lambert's pure iron and lead were prepared with a view to eliminating both these factors. Neither metal was incorrodible in certain conditions. In fact, the effect of strain is a minor and ephemeral factor in corrosion in neutral solutions.

As regards the effect of impurities on the corrosion of metals a trace of impurity appears to assist local corrosion, but the amount of corrosion is not proportional to the amount of impurity. Even the presence of graphite does not appreciably stimulate the rate of corrosion of iron. The effect of a trace of impurity is probably a trigger action.

Local action at metallic surfaces may be produced in a variety of ways at any selected points by modifying the conditions external to the surfaces, and is not mainly determined by the presence of anodic areas on the metal. Minute pores in a metal may, however, give rise to local action, as has been shown by Seligman and Williams.

On the electro-chemical theory the action of oxygen is that of a depolarizer. It can be shown, however, that atmospheric oxygen has very little depolarizing power at ordinary temperatures. The main function of oxygen in corrosion is to oxidize directly the metal, and also in some cases the products of corrosion.

Two chief types of corrosion are distinguished: (1) The general type usually characteristic of acid corrosion; (2) the local type, usually characteristic of corrosion in water and salt solutions. The second type is generally characterized by the formation of an adherent scale on the metal, and this scale may contain colloid.

A theory is developed regarding the part played by colloids in corrosion, and this theory may be briefly outlined as follows:—

A metal immersed in water sends positively charged metal ions into the liquid, and becomes itself negatively charged. In the case of ordinary commercial metals, the metal also

becomes superficially oxidized if dissolved oxygen is present. The hydroxide produced by this oxidation can take up the ions given off by the metal, and the hydroxide thereby passes into the state of a positively charged colloid. Some of this colloid will diffuse away, permitting further reaction between the oxygen and the metal surface, and thereby reforming the hydroxide film over the latter. Oxidation is then stopped till this hydroxide can pass into the colloidal state by acquiring positively charged metal ions. This, in general, does not take place till the colloid initially formed has diffused into the presence of electrolyte, when it is precipitated by the anion of the dissolved salt, the action neutralizing the charge on the metal corresponding to that on the colloid. This allows the metal to send more ions into solution, and the uncharged hydroxide thereby acquires a charge. If the colloid so produced can diffuse away, the process can continue and corrosion develop.

For steady corrosion, therefore, the colloid must be produced under conditions which allow it to diffuse some distance from the metal before precipitation. If it precipitates directly on the corroding surface it will, in general, adhere to the latter and stop corrosion. In the case of a corrosion pit, the first condition is fulfilled, since no precipitation occurs inside the pit. It is only when the colloid diffuses through an aperture (generally very small), in the gel-deposits at the mouth of the pit, that it meets electrolyte and is then precipitated. Such precipitation merely thickens the external gel-deposits. These gel-deposits adhere directly to and protect the metal surrounding the pit, and thereby emphasize the local nature of the corrosion.—G. D. BENGOUGH and J. M. STUART, *Inst. of Metals Reports*, September 20, 1922. (H.A.W.)

THE MANUFACTURE, USES AND CHEMISTRY OF EUCALYPTUS OIL.—The genus *eucalyptus* is indigenous to Australia, Tasmania and New Zealand, and its trees form three-fourths of the whole vegetation of this continent. Its immense importance can only be appreciated when we consider the value of its timbers, tannins, essential oils, etc., and in thinking of the eucalypts these other factors must not be overlooked, for the production of oil is but a small part of its uses.

The number of known species of eucalypts is about three hundred, and over 80 per cent. of them yield good timbers, including the well-known jarrah and some other of the finest hardwoods in the world.

Although three hundred species of eucalypts are known, only about twenty-five of these are used for the distillation of oil. Victoria distils by far the greatest amount of oil of any of the States.

The oil is extracted from the leaf by steam distillation, and many of the plants used are very crude in construction, consisting of one or more 400-gallon tanks. A frame, a few inches in height, is placed inside the tank, and on this the leaf is packed. The tank is then about half filled with water, and the lid placed in position and securely fixed by flanged joints clipped tightly together. A fire is lit beneath the tank and the steam and oil vapour are led off, condensed and the oil separated. These

crude plants have some advantages, especially if it is preferable to move the still instead of carting the leaf to it. The tank holds approximately 7 cwt. of leaves, and the yield of oil, varying, of course, with the species, nature of the leaf, etc., is usually about 15 lbs. from one charge.

In contrast to these crude plants there are many up-to-date ones working on a large scale and employing mechanical devices for the loading and unloading of the still. Several such plants are working on Kangaroo Island. In one particular plant inspected the steam was generated in a boiler and blown into the digester which contained the leaf. This digester was really a cemented pit about 8 feet in diameter and 8 feet deep, and it held about two tons of leaves and twigs. The digester was loaded and unloaded mechanically, thus effecting a considerable saving in time and labour. The steam was introduced into the bottom of the pit, and together with the oil vapour came over through a still head and was condensed in the usual way. One man, attending to the still only, could easily distil more than twice as much oil with this plant than he could working with the ordinary pot still. The leaves from which the oil had been extracted were used as fuel, and was more than sufficient to generate the required amount of steam.

The crude oil which is thus produced has to be refined if it is intended for medicinal purposes. Different types of oil, of course, require different treatment, and it is often necessary to distil in the presence of some chemical body to remove the undesirable constituent. Such methods are more or less secret to the trade.

Certain eucalyptus oils have been successfully used in the flotation process for the separation of metallic sulphides from the ore. Oils containing phellandrene and piperitone, such as those of *E. phellandra* and *E. dives*, have proved very satisfactory for this purpose.

Several eucalyptus oils have found extensive application in the perfuming of cheap soaps and in the preparation of perfumed spirits. The oil of *E. Macarthuri* is the one most used for the latter purpose. This oil consists mainly of geranyl acetate, with a small quantity of the alcohol geraniol. In the future it is quite possible that thymol and menthol will be prepared from piperitone, which is a constituent of several eucalyptus oils.*

Many eucalyptus oils are largely used in pharmacy, the principal constituent of these oils being cineol. The British Pharmacopoeia requires that oils intended for internal use shall contain not less than 55 per cent. cineol.

Messrs. Baker and Smith have divided the principal cineol-bearing oils into two main groups. The first group, of twenty-one species, includes oils consisting principally of cineol and pinene, the cineol exceeding 55 per cent. Phellandrene and aromadendral are absent from these oils. Of these twenty-one species the three principal ones used for the commercial production of oil are *E. globulus*, *E. Smithii* and *E. Australiana*. The second group contains ten species, which yield oils rich in cineol, but in which pinene is diminishing and aromadendral is appearing. From this group five species

are exploited commercially for the oil, *E. dumosa*, *E. oleosa*, *E. odorata*, *E. cueorifolia* and *E. polybractea*. These species are mallees, that is, they occur more in the form of a shrub than a tree, and thus it is easy to collect the leaves. *E. polybractea* is largely used in Victoria and in parts of New South Wales, while *E. odorata* and *E. cueorifolia* are the principal oil-yielding species of South Australia.

By far the greatest amount of South Australian oil is produced from *E. cueorifolia*, this species being about then three times as abundant as *E. odorata* on Kangaroo Island.

The best results are obtained by cutting the leaf, then rolling and scrubbing and planting a crop such as barley. After the crop has been cut the land is left and the eucalypts grow very quickly. In three years' time from the previous cutting the leaf is again ready to cut.

Eucalyptus oil, when exposed to air and light, increases in specific gravity, refractive index and dispersion, and diminishes in rotation. The specific gravity of a freshly distilled oil was 0.910, and after eighteen months' exposure to light in a white glass corked bottle, the gravity had increased to 0.930.

The bacterioidal value of a substance is usually estimated with respect to *bacillus coli communis*, and when this is done for eucalyptus oils the result is not as good as might be expected. Eucalyptus oils have very low germicidal values with respect to *B. coli communis*, but from their efficacy in colds, etc., there is little doubt that they would appear much higher in the scale if tested with organisms associated with colds. With reference to the therapeutic action of eucalyptus oil, competent authorities state that it is by no means certain that the therapeutic action is due to the cineol content, and it seems probable that the medicinal value is more largely due to the admixture of certain constituents than to that of any one constituent.—P. A. BERRY, *Chem., Eng. and Min. Review*, May 5th, 1922, pages 262 and 263. (H.R.A.)

METALLURGY.

DESCRIPTION OF REDUCTION PLANT OF TRANSVAAL SILVER AND BASE METALS MINE.—The plant capacities are as follows:—

	Tons per month. (2,000 lbs.).
Crusher and sorting station	7,500
Concentration plant	6,000
Smelting plant (production silver- lead bullion)	500

Crusher and Sorting Plant.

The crusher and sorting station is of the ordinary Witwatersrand type, with receiving bin, primary jaw crusher (crushing to 5-inch cube) by belt conveyor to trommel and washing plant; then over sorting belt to two secondary crushers, and on by belt conveyor to the mill storage bin of concentrate plant. All fine rock screened by the headgear grizzlies is also trucked to this conveyor belt for delivery to the mill bin.

The sorting floor is carried over the waste bin, but after the waste has been sorted a

* Paper by H. G. Smith, in the Proceedings of the Royal Society, N.S.W., 1920, page 40.

separate bin is provided, in which all high-value hand-picked ore is placed. This ore can be sent direct to the roasting plant without concentration. The waste sorted out is weighed and trucked to a waste dump.

The two secondary crushers are of the Sturtevant roll jaw type, size 20in. by 6in. These are of an old type, but were purchased on the Rand, having never been in use. Each crusher, when breaking to $\frac{3}{4}$ in. mesh, is capable of dealing with from ten to twelve tons of ore per hour. All ore, except screened fines—minus $\frac{3}{4}$ in. mesh—is passed through them after sorting and hand picking.

Concentration Plant.

For this plant it was decided to adopt the Mineral Separation process for treatment of ore.

If it had been decided to ship concentrates to Europe, and not to smelt locally, the design of this plant would probably have consisted (as in many base metals plants in the States) of mill bins, ball mills, "all-sliming" everything, and, finally, the flotation unit.

In view, however, of the difficulties of desulphurising and smelting the fine concentrates from the flotation process without mixing it with coarse concentrates, it was decided to erect rolls, gravity concentration plant, with jigs, before passing the ore to the ball mills and fine grinding for flotation.

In designing this plant, the first consideration was the launder grade to be adopted, the gravity of the milled pulp being about 4.0, as against 2.7 for ordinary banket. After careful consideration it was decided to make the grade of the coarse sand launders 30 per cent., with a minimum grade for the finer sand of 10 per cent.

These grades seem heavy as against ordinary Witwatersrand practice, with a maximum grade of about 12 per cent. for heavy pulp, but from experience in working I may say that the coarse $\frac{3}{4}$ in. mesh pulp will not run on 30 per cent. grade without excessive water.

The general arrangement of this plant is as follows: From the main 250-ton mill bin the ore is delivered by two ratchet-driven variable stroke worm feeders to the boot of the No. 1 bucket elevator. This elevator is equipped with 56 buckets, each of 300 cubic inches capacity, running at 310 feet per minute, and delivers on the first floor of the building into a $\frac{3}{4}$ in. mesh, sizing and washing trommel. The oversize, from this trommel is delivered over a short conveyor belt with a magnetic head pulley, to remove any tramp iron, to a set of 16in. by 36in. Allis-Chalmers rolls. After passing the rolls, which crush to $\frac{1}{4}$ -inch, the product returns to the boot of the No. 1 elevator, thus making a closed circuit, the only ore leaving which is that screened through the wash trommel as minus $\frac{3}{4}$ in. product. This $\frac{3}{4}$ in. product falls to the boot of No. 2 elevator, containing forty buckets, and delivering to the second floor of the building, where, after sampling for assay value, it is passed through two sizing trommels with $\frac{1}{2}$ in. mesh screen.

These are the first sizing trommels for six two-compartment Harz jigs, each with a screen area of 11 sq. ft., which are placed on the jig floor. The plus $\frac{3}{4}$ in. product from the trommels passes to numbers 1 and 2 jigs, the undersize with the pulp passing to a hydraulic

classifier, which sizes the product for the numbers 3 and 4 and numbers 5 and 6 jigs. The fine pulp overflow passes to the ball mill head cones.

There is no reject tails from the jigs, the mineral in the ore being very finely disseminated; the whole overflow from the jigs is therefore passed to the ball mill cones.

The concentrate, either from the jig screen or hutches, is collected in conical bins, and after discharge is trucked to the roasting plant.

The water used in this plant is considerably more than in gold milling practice, and has at the ball mill cones reached a total of nearly 50,000 gallons per hour, or 250 tons of water carrying 10 tons of ore per hour—i.e., 25 to 1 ratio.

This water is added to the pulp, first at the rolls and elevator, then in the three trommels and classifier, and, finally each jig requires from 4,000 to 5,000 gallons of water per hour to obtain a good clean concentrate.

The overflow of the ball mill cones is run direct to a 40ft. Door thickening tank, and the water, after settlement, is returned to a 60,000-gallon steady-head tank, situated above the concentrate plant.

The underflow of the cones is taken to two Door classifiers in closed circuit with two Hardinge 6ft. by 22in. conical ball mills, 2in. diameter steel balls being used in these as the grinding medium. Each of the mills is capable of grinding 100 tons per day, the average grading at present being:—

Classifier intake—60 per cent. plus 0.1in.;
32 per cent. plus 0.01in.; 8 per cent.
minus 0.01in.

Classifier outlet—12 per cent. plus 0.006in.;
29 per cent. plus 0.003in.; 59 per cent.
minus 0.003in.

The classifiers were originally erected with a grade of 2in. to the foot, but, owing to the pulp gravity, this has now been increased to 3in. to the foot, and a worm conveyor installed to deliver the classifier product to the ball mill feeder.

The consumption of steel balls on these mills, each grinding 100 tons per day on the above grading of pulp, is about 1.5 lbs. per ton ground.

To the ball mill circuit is added the tar required for the flotation process, this being done in order to get it intimately mixed with the finer particles of mineral in the ground ore. The tar at present used is obtained from the Johannesburg Municipal Gas Works.

The Dorr classifier overflow is delivered on to fine screening to remove as far as possible any wood pulp or coarse chips which tend to choke the flotation machine, and after passing this screen is delivered to sands pumps for elevation to a main cone in the flotation house, the overflow of this cone passing to the Dorr thickener.

In the flotation house the underflow of this cone is laundered to the head of a 10-compartment Mineral Separation Company sub-aeration type machine, and to the same launder is added the fine slime from the Dorr thickener, which is raised by a 4in. diaphragm pump, also the remaining flotation re-agent, consisting of cresol with a water ratio of, roughly, 3 to 1.

The type of flotation machine in use consists of a series of ten compartments, each about 18 inches square, built up in the form of a box launder 25 feet long by 2 feet wide by 3 feet deep from the discharge lip. In each compartment a propellor revolves, driven from an overhead shaft by bevel gearing at a peripheral speed of about 1,000 feet per minute.

Air is supplied by a small blower of 100 cubic feet capacity, at about 2 lbs. pressure per square inch, and is admitted to each compartment below the centre of the propeller, and the compartments are covered with wooden grids about 15 inches below the pulp level. The air, broken into small bubbles by the propeller, rises through the pulp and floats over the lip of the machine as a froth, carrying any mineral particles which are attached to the air bubbles by surface tension.

The first compartment of the machine is used for agitation only; from the next five compartments a high-grade finished concentrate is obtained; on the last four compartments of the machine a low-grade concentrate is obtained, this being returned to the head of the machine for re-agitation and re-treatment.

The high-grade concentrate, after breaking down the froth with a water jet, is pumped to settlement cones, and from these trucked to pits for sun-drying. The tailings from the machines are pumped to sand and slimes dams.

The plant was started up in January, 1922, and the cost of operation and extraction results have reached the figures originally estimated.

The jig concentrate, containing about 60 per cent. of the total ore value at the head of the plant, is a clean smelting product containing an average of 60 per cent. lead and 10 per cent. sulphur. The mineral separation concentrates are not quite so high in lead, and the sulphur contents is in the neighbourhood of 15 per cent.

The whole of this product is trucked to the desulphurising plant, also any hand-picked galena from the crusher station.

Desulphurising Plant.

The resulphurising plant in which the concentrates are next treated consists of three Huntingdon-Heberlein pots of 9 feet diameter by 3 feet deep, each 160 cubic feet capacity, holding about 12 tons of charge. The pots are mounted on a carriage for tramping to the breaking floor, and carried on a pivot, so that the contents, when desulphurised and sintered, can be discharged on the floor.

All concentrates passing 4 in. mesh are bedded on a mixing floor with granulated slag, ironstone and limestone, all crushed to 4 in. mesh, and after a thorough mixing delivered by a conveyor belt to the bin in the pot house.

The pot house and blast furnaces are supplied with air for blast purposes by three Roots blowers, each capable of delivering 3,000 cubic feet of free air per minute against 2 lbs. pressure per square inch. These blowers are housed in a separate house, situated between the pot house and furnace plant.

The method of operating the desulphurising pots is as follows:—

A coal fire is lighted on top of the wind-box

grate, and air is supplied from the blower mains. A layer of mixed charge is then placed over the hot coals, and the sulphur in the ore starts to burn. After a 4 in. layer of the charge is well lighted, the pot is filled with mixed concentrate charge from the bin, and the air blast is turned on full with about 8 oz. air pressure.

After a 10-hour roast the majority of the sulphur contents is burned off, being reduced to about 3 to 4 per cent., and the heat given off by the same has sintered the mass into one solid cake. This caked sinter is then tipped from the pot on to a breaking floor, and is broken up into 4 in. to 6 in. cubes for smelting in the furnaces.

The sulphur fumes from the pot plant are collected by hoods placed over the pots, and are discharged into the main flue, which is 250 feet long, 6 feet by 6 feet inside diameter, with a 7 feet diameter by 100 feet steel chimney. Into this same flue the blast furnace gases are discharged after passing through a dust settling chamber.

Smelting Plant.

In the furnace house two blast smelting furnaces are installed, one a small circular furnace of 36 in. diameter, which was imported, and the second a rectangular blast furnace 6 feet by 3 feet at the tuyeres, which was completely built in Johannesburg. Both furnaces are of the water-jacketed type, with a 20 ft. charge stack between the crucible and the charge floor.

The water jackets of the rectangular furnace, weight 4½ tons, were built and electrically welded locally by the Hydrogen Oxygen Plant Company, and up to the present, after four months' run, these jackets have given no trouble of any sort.

The furnaces are charged from the charge floor, at the back of which, carried on concrete piers, are situated the ore and flux bins, containing sintered and oxidised ore, ironstone, limestone and the necessary coke. All the charge is elevated to these bins by means of an electric hoist, which winds trucks up an inclined plane.

Each furnace charge at quarter-hourly intervals is weighed on the charge floor and is fed into the stack of the furnace. The blast required for smelting is from 3,000 to 5,000 cubic feet of air per minute at about 20 ozs. pressure.

The lead is run off from the lead well in the crucible by means of a syphon built in the crucible. The slag and matte is tapped at the end of the furnace, and after parting the slag is placed on the dump and the matte containing any copper in the ore is re-crushed and again passed through the roasting process, to eliminate the sulphur.

The silver-lead bullion from the lead well is run into 110 lb. pigs, and in this state is shipped to Europe for softening and desilverising. The amount of silver obtained from the pig at present smelted is about 100 ounces to the ton of lead.—H. W. CLAYDEN, *The Journal of the South African Inst. of Engineers*, September, 1922, p. 20. (F.W.)

MINING.

AIR-COOLING AT ST. JOHN DEL REY.*—During the hot and rainy season, when the absolute moisture-content of the surface air entering the downcast shaft was almost continuously above its average value, the absolute moisture-content of the air passing any point underground was also continuously above its average value for that point; hence, since the dry-bulb temperature and barometric pressure at any one point underground remained approximately constant, the wet-bulb temperature at that point was also continuously above its average value. During the dry and cooler winter season, when the absolute moisture-content of the surface air entering the downcast shaft was almost continuously below its average value, the wet-bulb temperature at any one point underground was similarly continuously below its average value for that point. As regards the stopes, the maximum wet-bulb temperatures attained were so high that while they were liable to occur it would not have been possible to work the mine profitably at depths much greater than those which had already been reached; however, during the winter season conditions in the stopes were considerably improved on account of the lower wet-bulb temperatures prevailing there. It was consequently decided to instal on the surface a plant through which should pass the whole of the air entering the mine, except that entering in a compressed condition by the air-compressors and air-mains to operate underground hoists, rock-drills, etc., and that this plant should continuously reduce the air passing through it to a condition approximately equal to that obtaining on a cold winter's morning. The guaranteed capacity of this plant was that it should have a heat-extraction rate of not less than 100.630 B.ThU. per minute; this corresponds to the reduction of 43.2° F. from an initial wet-bulb temperature of 72° F. and any degree of saturation of 5,040 pounds of dry air per minute.

The plant is similar in principle to an ordinary cold-storage plant with a brine-circulation system, but as the temperatures do not fall below 32° F., ordinary water is used instead of brine, and the place of the cold-storage chamber is taken by two large air-coolers. As the wet-bulb temperature of the air entering the plant is liable to have any value between 75° and 32° F., according to the season and time of day, and as it is obviously desirable to supply the mine with air at a fairly steady temperature, the plant is divided into six stages, in which the temperature is reduced respectively from 72° to 67.6° F., 67.6° to 62.6°, 62.6° to 57.6°, 57.6° to 52.6°, 52.6° to 47.8°, and 47.8° to 43.2°. Hence, when the wet-bulb temperature of the air entering the coolers is between 67.6° and 62.6° F., the first stage can be shut down and the plant run on the remaining five stages; when it is between 62.6° and 57.6°, the first and second stages can be shut down, and so on, until, when the initial wet-bulb temperature is below 43.2°, as happens on cold winter nights and mornings, the whole plant can be temporarily put out of commission.

When in December, 1920, the plant was put into regular twenty-four hours service, it was purposely run for some months with an average exit-temperature slightly over 50° F.; however, during the cold months (May to August, 1921) the initial temperature of the air was, during the night, frequently well below this figure, and thus the average cooling-plant exit-temperatures for these months also fell below it. At the end of November, 1921, after practically a year's running, the exit-temperature was lowered to approximately the figure for which the plant was designed (43° F.), where it has since remained.

A marked dry-bulb temperature-drop has taken place throughout the whole of the downcast way, and the air now reaches Level No. 22 at a temperature 3.7° F. lower than that at which it used to arrive at Level No. 21. In the stopes (working zone) the drop is not so great, owing to the fact that fresh and comparatively uncooled rock is always being brought into contact with the air; even so, however, the average reduction for selected points is 2.5° F. Higher up in the upcast the drop becomes more considerable, owing probably to heat-leakage from upcast to downcast through the intervening rock. The sudden rise in temperature (5.9° F.) which the upcast air now experiences at Level No. 14 is due to the combined effects of compression and friction-loss in its passage through the tandem fan, and shows how important it is that such fans, when used in a hot mine, should not be placed in a position where the air discharged from them passes immediately over any place where miners are at work.

The relative humidity of the ventilating current decreases steadily at the present time from 100 per cent. on exit from the cooling plant all down the downcast, reaching Level No. 22 with a value of 37 per cent. After reaching the minimum value of 36 per cent. it begins to rise rapidly, 100 per cent. being again attained at about Level No. 8.

It is estimated that with a mean wet-bulb temperature in the stopes of 90° F., the average wet-kata cooling power obtaining there was 7.7 millicalories per second per square centimetre, with maximum and minimum values of 9.1 and 6.1. As the figure for wet-kata cooling power as being the minimum compatible with efficient working may be taken at 16, it will be seen that from this standpoint the stopes at Morro Velho were liable before the cooling plant started up to get into a very bad condition indeed, and it is probable that they never reached a satisfactory figure at any time during the summer months. At the present time, however, the average wet-kata cooling power on the stopes (excluding one abnormal point) reaches the high figure of 20.5, the maximum and minimum values being 24.3 and 16.8 respectively. The cooling plant cannot, of course, claim all the credit for this enormous improvement, for cooling power as measured by the kata-thermometer is dependent on air-velocity as well as temperature, and the velocity in the stopes, in common with that throughout the mine, is now 40 per cent. higher than it used to be, for reasons already given.

* See this *Journal*, Vol. XXI., November, 1920, p. 102.

Another improvement directly due to the operation of the cooling plant is that which has taken place in the condition of the air blown by the auxiliary ventilation fans into the working-faces of the dead-end development drives at the lowest level. In support of this it may be mentioned that whereas wet-bulb temperatures between 91° and 92° F. were frequently observed in the dead-ends at Levels No. 22 before December, 1920, nothing higher than 84° F. has been read during the last six months. This is naturally a result of great importance, since the extension of the workings in depth depends entirely upon these drives and the speed at which they can be pushed forward.—E. DAVIES, *The Mining Magazine*, from paper read before the Institution of Mining Engineers, July, 1922, page 49. (C.J.G.)

NOVEL MAN CAGE.—The Seneca shaft of the Seneca Copper Corporation, Calumet, Mich., has two compartments, with a ladderway in the centre. The first 1,320 feet, is vertical. The shaft then passes through an arc of 610 feet on a 600-feet radius and into a slope at 32 deg. for 770 feet further. The curved part is provided with twenty 4-feet sheaves in each compartment.

In order to handle both ore and men in such a hoistway and keep the skips at all times vertical, a specially designed skip has been devised. This skip, which dumps like a Kimberley, has proved successful.

The man skip is not unlike a single cage of a Ferris wheel, suspended on trunnions between two vertical steel beams. The top is rounded to avoid hitting the roof, on the curved part of the shaft, and the entire cage is covered with heavy wire net. To reduce oscillation when in motion, and owing to change of direction, the pantograph side bars are attached to oil cylinders, which act in the same manner as shock absorbers on an automobile, and adjusted to any given load and speed. As a result, the cage passes through shaft, curve, and incline as steadily and as normally vertical as though continuously in a straight shaft.

The drifts are 8 feet by 12 feet in dimension, with a 24-inch gauge track. Six-ton cars on two four-wheel trucks are used for tramping, the body being 16 feet long and 4 feet 4 inches wide, with drop doors. The axles of the trucks are 30 inches wide centre to centre.—*Queensland Mining Journal*, October, 1922, p 417 (J.A.W.)

FIRST AID TREATMENT FOR CO-POISONING.—The Bureau of Mines U.S.A. has recently issued a report of investigation on the treatment of CO-poisoning. Although a frequent cause of industrial accidents, there appears to be no uniformly recognised treatment for a person overcome by this gas. The treatment recommended by the Bureau is as follows:—

1.—Administer oxygen as quickly as possible in as pure a form as is obtainable, preferably from a cylinder of oxygen through an inhalator mask.

2.—Remove from atmosphere containing CO.

3.—If breathing is feeble, at once start

artificial respiration by the prone pressure method.

4.—Keep the victim flat, quiet and warm.

5.—Afterwards give plenty of rest.—*Chemical and Metallurgical Engineering*, January 25, 1922, Page 179. (J.A.W.)

MISCELLANEOUS.

THE EFFICIENCY ENGINEER AND THE INDUSTRIAL PSYCHOLOGIST.—The author points out to the efficiency engineer the limitations of his work unless it is carried out with due regard to its psychological complications. The efficiency engineer commonly proceeds by analysing a given operation into a number of different parts, observing and tabulating the movements of those operatives who perform each of those parts in the quickest time, finally collecting into one series the quickest and best movements as the "one best way," which is then stereotyped and forced on every worker. To this the trained psychologist retorts that the psychological and physiological differences between individual workers are such that it is impossible to train and expect all to adapt themselves to perform operations in precisely the same way. Nor is it justifiable to expect that the speediest and shortest movements are necessarily the least fatiguing. Furthermore, laboratory tests carried out with ergographic and mental work tests are not directly applicable to actual working conditions. They can only assist, but never solely determine an appreciation of any individual's special aptitudes. In determining and setting the task which may be expected of an efficient worker the industrial engineer often does not appreciate the various emotional influences and incentives which affect the worker's efficiency. The author points out that whilst there are many improvements in industries which are independent of physiological and psychological knowledge and can best be carried out by the efficiency engineer, there are others which devolve on the industrial psychologist, either alone or in co-operation with the efficiency engineer.—CHARLES S. MYERS, *S.A. Mining and Engineering Journal*, Abstract of Paper read before the British Association, 14th October, 1922, pp. 99 and 100. See *Journal of this Society*, Vol. xxiil., August, 1922, p. 20. (H.P.)

CHANGES OF ADDRESS.

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PROUT, W. M. 1/o Lutzputs, via Upington, C.P.; Leonie, Pienars River, District Pretoria.

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THE LOWEST TEMPERATURE YET ATTAINED.

By H. H. PAINE.

The arrival on the Rand of a machine for the manufacture of liquid air (or, more correctly, liquid oxygen) makes it of special interest to enquire into the limits of low temperature work, particularly when a new record for the lowest temperature obtainable has been reached quite recently. In October last a general discussion was held before a joint meeting of the Faraday Society and the British Cold Storage and Ice Association on "The Generation and Utilisation of Cold," and at that meeting Professor Kamerlingh Onnes read a paper "On the lowest temperature yet obtained," in which he gave a preliminary account of experiments he had recently been carrying out at Leiden. The following article is practically a resumé of that paper:—

The importance of the study of low temperature and its effects on the properties of matter of all kinds has led to the establishment of a Cryogenic Laboratory at Leiden designed solely for the study of these phenomena. The main purpose of this institution is to be able to produce any temperature below zero, and (which is more important and much more difficult) to maintain such temperatures to within one-hundredth of a degree continuously for several hours. Side by side with the carrying out of this purpose experiments have been conducted to see what is the lowest temperature that can be reached.

The principle made use of at Leiden, in every case where a low temperature has to be maintained, is that at any given pressure every pure liquid has a fixed and constant boiling-point. Generally, where low temperatures are sought, the liquids will be

made to boil at low pressures and hence a suction (gas) pump is an important feature of the apparatus employed. At these low temperatures heat enters the liquid from its surroundings, either by radiation or by conduction. Since the temperature is to be kept constant this heat must be used up in converting some of the liquid into vapour. The function of the pump is to remove this vapour as it is formed, and hence, in maintaining a constant temperature, everything depends on the efficiency of the pump and the regularity with which it can be kept going. If now the pump is worked faster, in order to remove the vapour more quickly, the pressure will be reduced and this will result in an increase of the rate of evaporation from the liquid and hence the temperature of the liquid will fall.

LIQUID HELIUM.

The last substance to be liquefied, that is the substance with the lowest boiling-point, is the element helium, first liquefied at Leiden in 1908. Its critical temperature is -267.84°C. , and at normal atmospheric pressure it boils at -268.83°C. It is interesting to note that at these temperatures no substance other than helium can exist in the liquid state. Hydrogen solidifies at -259°C. So far helium has never been solidified, and it is on account of this fact that the attempt to reach lower and lower temperatures becomes possible on the above lines. Solids have definite pressures, it is true, but to obtain a "bath" of the solid at a uniform homogeneous temperature by a control of the pressure would be practically impossible. Since helium remains

liquid at the lowest temperatures reached the limit to these temperatures depends almost entirely on the efficiency of the pumping arrangement.

In the experiments to be described considerable amounts of helium were used and consequently such experiments only became possible after the war, during the course of which large quantities of the gas had been collected in the United States and in Canada.

The helium was carefully purified and then liquefied by the Linde process, making use of the Joule-Thomson effect. As in the case of hydrogen, this is a "heating" effect at ordinary temperatures, but an inversion temperature exists, and below this free expansion through a nozzle produces a cooling of the gas. The Linde process therefore necessitates the initial cooling of the helium by liquid hydrogen before its cycle of operations can be applied.

From the Dewar flask in which it was prepared the liquid helium was syphoned over into an apparatus designed for reducing the pressure over its surface as much as possible. This consisted essentially of an evaporation flask, containing the liquid, which was connected to a powerful vacuum pump.

Since the object of the experiment was to read as low a temperature as possible, the pumping was made very efficient so as to reach as low a pressure as possible. As we shall see later, the temperature of the liquid had to be deduced from its vapour pressure; hence special means were devised for measuring this pressure. Finally, since the temperature was very low, special precautions were taken to insulate the evaporation flask and so to prevent heat from reaching the liquid helium which would cause it to boil away rapidly. The special arrangements designed for attaining these objects make this experiment important, and it is with a brief account of them that we shall be concerned in the next few paragraphs.

THE PUMPS.

The main pumping was performed by two large and powerful Burckhardt vacuum pumps in series. Between these and the evaporation flask was set a battery of eighteen of the new Langmuir mercury vapour condensation pumps arranged in parallel. Such an arrangement resulted in a vacuum pump system of very large capacity and extremely low suction pressure.

All tubes between the evaporation flask

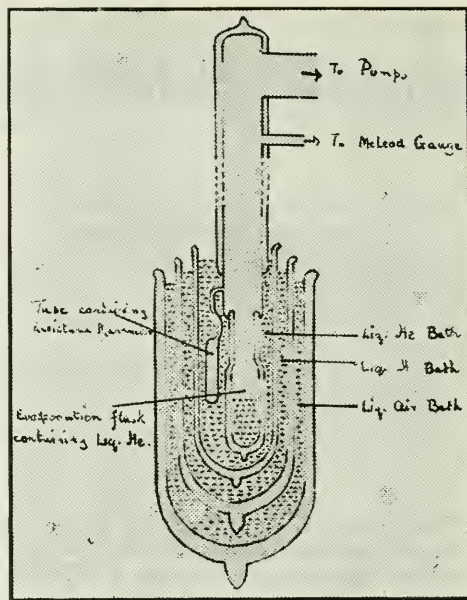


Diagram to show the principle of the method used.*

and the pumps were as wide as possible. At very low pressure the mean free path of the molecules becomes long, and the frictional resistance that arises when the low pressure vapour is passing through the tube creates a measurable difference of pressure between one end of the tube and the other. Hence the necessity for wide tubes when exhausting to very low pressure in order to cut down this frictional resistance. In Onnes' experiment the main tube was 12 cm. in diameter.

THE ABSORPTION OF HEAT.

As the temperature of the liquid helium was so low every precaution possible had to be taken in the endeavour to prevent outside heat from reaching it. This was all the more important in that the latent heat of vaporisation of helium is only 6 calories per gram, and therefore very little heat is required to make the liquid boil so rapidly, that the pumping, however efficient in more normal circumstances, would not keep the pressure above the liquid down to anything very small.

Wherever possible, in all parts of the apparatus where the temperature was low, the vessels and tubes were double-walled with evacuated interspaces. Around the

* This diagram is intended merely to illustrate the present article, and omits many of the important features of the actual apparatus on which the success of the experiment depended.

evaporation flask itself several consecutive vessels of this type were constructed, one outside the other. The one next to the evaporation-flask contained liquid helium, the next liquid hydrogen, and the next liquid air. Around this outermost vessel a bath of alcohol, kept at ordinary temperature, was placed to prevent the deposition of moisture which would render impossible the observation of the inner vessels. The tubes leading to the pumps were made of glass as thin as possible, to avoid conduction along the walls of the tubes from regions of ordinary temperature to the evaporation flask. Even the conduction along the column of low pressure helium vapour had to be taken into account, and set a limit to the cross section of the tubes through which this vapour had to pass from the flask to the pumps.

Radiation is important at such temperatures. The total radiation from 1 sq. cm. of a surface at ordinary temperature to a region at -270° C. can be shown to be 30 gram-calories per hour—sufficient to evaporate 5 grams of the liquid helium. Hence everything had to be surrounded by opaque (metallic) screens kept at a low temperature by means of liquid helium or liquid hydrogen. Great skill was exercised in the construction of the glass apparatus to enable such suitably cooled screens to be inserted at important points. The Dewar flasks around the evaporation flask were all silvered, except for a pair of narrow slits which enabled observation of the innermost vessel to be maintained. A movable, metallic screen in the helium bath closed these slits when observation was not required.

When all such precautions were taken, the evaporation was reduced to about one-sixth gram of helium per hour, or 1.12 c.c. of the liquid.

MEASUREMENT OF THE PRESSURE.

As will be explained later, in order to determine the temperature, it was necessary to measure the pressure over the surface of the liquid helium—that is, the vapour pressure of the helium. This presented two difficulties. (1) Very low pressures are usually measured by means of a McLeod gauge. But in this case the pressure to be measured was that over the liquid helium at a temperature of about -270° C. The temperature of the manometer had to be somewhere about atmospheric. At such low pressure this difference of temperature between the gauge and the evaporation flask

in itself brought about a difference of pressure between these two regions—the thermal molecular pressure. Hence, the pressure recorded by the gauge would not be the pressure that it was desired to measure. (2) Again, there was the difficulty which arose from the fact that the helium vapour at low pressure was moving rapidly from the evaporation flask to the pumps. This brought about a difference in pressure between any two points in the direction of flow of the vapour, so that the pressure recorded at one point would not be the pressure at another point further along the tube. Structural difficulties prevented the fixing of a wide enough manometer tube from near the level of the liquid helium to the gauge.

As a consequence a new arrangement for measuring low pressures was employed, known as a resistance manometer. Suppose we have a platinum wire in a perfect vacuum heated by an electric current. The wire will increase in temperature, until the heat lost by radiation (alone) just balances the heat generated by the current. If now there is a small quantity of air or other gas present, the wire will lose heat not merely by radiation as before, but also by molecular convection or conduction, and this extra loss will depend on the pressure of the gas. If the current through the wire is increased until the temperature of the wire is the same as before (that is, until its resistance, which increases with temperature, is the same as before), and the surrounding temperature also remains the same, the heat lost by radiation will be the same as before, and the excess heat produced (calculated from the increase in the current) will be the heat carried away by the gas. It would be impossible in practice to *calculate* the pressure of the gas from the heat carried off by the gas, but such an arrangement can be *calibrated*, and when once this calibration is made the resistance manometer can be employed with certainty.

The resistance manometer was placed in a small tube leading out of the evaporation flask at a point close to the surface of the liquid helium. The tube itself was in the bath of liquid helium which surrounded the evaporation flask.

The calibration consisted of a comparison with the readings of the McLeod gauge carried out when the above-mentioned difficulties did not interfere. There was still a liquid helium bath around the evaporation

flask and the manometer tube, but there was no liquid helium in the flask, and therefore the gas was at rest. The whole of the tube connecting the main part of the apparatus to the gauge was at atmospheric temperature, and the fall of temperature occurred along the wide straight tube above the flask. In these circumstances the thermal molecular pressure was small, and could be calculated approximately.

This then was the arrangement used for recording the vapour pressure of the liquid helium. In this way Onnes found the lowest pressure he could reach to be 0.013 m.m. of mercury. In 1910, when he carried out a similar experiment, with less efficient apparatus, the lowest pressure recorded was 0.2 m.m. of mercury.

TEMPERATURE.

The determination of the temperature presents some interest, for we are concerned with a region in which all ordinary methods break down. Temperature is usually defined by means of a gas (hydrogen) thermometer, and all thermometers in actual use ought to be calibrated by comparing them with the standard. But at such a low temperature as that which we are now considering hydrogen is a solid with an infinitesimal vapour pressure. A gas thermometer filled with helium is available to within two or three degrees of the temperature we wish to measure, but where the maximum vapour pressure is only a small fraction of a m.m. even a low pressure helium thermometer is not available.

Again, anything in the nature of a resistance thermometer is impossible here. At the temperature of liquid helium platinum has a very low resistivity which does not vary appreciably with temperature.

Hence we fall back on extrapolation methods based on the law of "corresponding states" of Van der Waal. If, for regions in which a (helium) gas thermometer is available for measuring temperature, we observe the maximum vapour pressures at various temperatures for a variety of substances; and if we calculate the ratio of the critical temperature of the substance to the actual temperature, and the ratio of the maximum vapour pressure at that temperature to the critical pressure; and if we then plot the first ratio against the logarithm of the second one we shall find that we get curves (one for each substance) which are approximately straight lines, from the critical point to the triple point of each substance, that is, for all temperatures at which

the substance exists in the liquid state. We can therefore extrapolate with fair confidence in the case of the helium curve, and thus deduce temperatures for observed values of the vapour pressure.

Before giving the results one other point must be mentioned. Gas thermometry suggests for us an "absolute zero." But there is something artificial about such a suggestion inasmuch as every gas ceases to be a gas long before this zero is reached, and all laws of volume or pressure change, therefore, break down. But the absolute thermodynamic scale of temperature devised by Kelvin is independent of the properties of any particular substance, and is based on certain fundamental "laws" (of thermodynamics). Hence it is "rational" in a way that gas thermometry cannot pretend to be. This absolute thermodynamic scale indicates the existence of an absolute zero at which all substances would be entirely devoid of heat, and hence could not undergo any further "cooling." The experimental gas (hydrogen) scale agrees closely with the thermodynamic scale (for temperatures well above the critical temperature of the gas), and the study of the thermodynamic properties of hydrogen (or of any other gas used to fill an actual thermometer) enables us to deduce the "absolute thermodynamic temperature" when the gas thermometer temperature is given.

Since absolute zero on the Kelvin scale signifies an absolute limit beyond which no experiment can take us, it is much more appropriate to express these extreme temperatures in terms of this scale, and in reference to this absolute zero. Using the extrapolations from the "corresponding states" curve for helium, a maximum vapour pressure of 0.2 m.m. reached in 1910 gave a temperature of 1.15° K. (that is, on the "Kelvin" scale*), while the maximum vapour pressure of 0.013 attained in 1922 gave the temperature of 0.82° K. This is the lowest temperature yet attained.

This is the final result obtained at Leiden to reach a temperature within five-sixths of a degree of absolute zero. Prof. Onnes estimates that, working along the same lines with helium, and improving the apparatus to enable him to obtain still lower pressures, it might be possible to descend another one-sixth of a degree. To go lower

* The "size" of the degree is still that of the centigrade scale, i.e., one-hundredth of the interval between the melting point of ice, and the boiling point of water at 760 m.m. of mercury pressure.

still, it would be necessary to devise some quite new method, making use of some other property of matter. A hundred years ago, when Faraday first liquefied a number of gases like carbon dioxide and nitric oxide, the lowest temperature reached was still 200° above the absolute zero, so that it would be unwise to set any limit to what might be attained even in the next decade.

It is not the purpose of this article to emphasise the importance of low temperature research. There are a number of questions awaiting answer in connection with the superconductivity of metals, and the extent to which such phenomena are general. But there is one which arises directly out of this paper which we might touch on. The element helium has never been solidified, although the absolute zero has been approached to within five-sixths of a degree.

The vapour-pressure of hydrogen at its solidifying point is 5 c.m. of mercury. The temperature of liquid helium has been reduced until its vapour pressure is less than 0.002 c.m. of mercury, yet it shows no sign of solidification. At the absolute zero matter is deprived of all its heat. On the Kinetic Theory we should then expect that the molecules would be at rest. It is difficult to conceive of the liquid state being possible at all if the molecules are not moving. We cannot help thinking, therefore, that the solidification of helium must set in before the zero is reached. It may be that in some way or another the distinction between solid and liquid ceases to hold at 0° K. In any case the "state" of helium at lower temperatures than have yet been reached very much concerns our interpretation of the Kinetic Theory.

RETURN TO CAPITAL INVESTED IN THE WITWATERSRAND.

By R. A. LEHFELDT.

The amount of the profits made by mining gold on the Witwatersrand is a matter of great importance to South Africa, and is of considerable economic interest to the world; but it is one on which there is very little information to be had, and so, naturally, very contradictory opinions have been held. The following paper is an attempt to work out the precise result over a certain period. It would be of more value if it went back to the beginning of the exploitation of the minefield (about 1887), but this appears to be almost impossible. It is doubtful whether records are still in existence that would provide the data necessary; in the early history of gold mining on the Rand almost innumerable ventures were started, with larger or smaller capital. In many cases money was spent on a piece of ground; it was abandoned, and subsequently taken up by another company or syndicate. Then amalgamation and financial rearrangements were frequent; it is difficult to make sure what was real and what only nominal capital; and so much information would be needed that it is almost beyond the capacity of a private investigator to handle it, even if the records exist.

The aim of the present paper is therefore restricted to the operations within a period

after the mine-field had become well established. The first point to be settled was the limits of the period chosen. It would give an erroneous notion to start or finish at a moment when the general feeling about the industry was unduly optimistic or pessimistic. In so far as financial estimates of the value of the mines are correct, they may rightly be taken into account, but it is well known that public opinion with regard to the value of mining enterprise—as reflected on the Stock Exchange—is very fluctuating and often erroneous. As a preliminary step a chart of the market price of a leading security—Rand Mines shares—was prepared, and by inspection of it, 30th June, 1907, was chosen as the initial date; that being a time when the Stock Exchange quotation seemed to be fairly steady and neither unduly depressed or inflated. As to the closing date it was difficult to find any sound reason for preferring one to another, so the latest available was chosen, viz., 31st December, 1921. It was desired to find what return on capital was made during this period of fourteen and a half years.

The method adopted was to suppose an investor, who, on 30th June, 1907, bought all the shares in the Witwatersrand mines

at the market price, who subsequently supplied all the new capital that was supplied, received all the dividends and repayments, and sold out his interest at market price on 31st December, 1921. What return would the imaginary investor have got on his money?

To carry out this plan involved examining a very large mass of records and accounts, as well as doing a formidable amount of arithmetic. The writer was fortunate enough to secure the active support of Sir Robert Kotze, Government Mining Engineer to the Union of South Africa, who placed the resources of his department at the service of the investigation. Thanks are due to the officials of the Department for their willing and interested labours, and especially to Mr. F. C. Vaughan. Without their help this piece of research would certainly not have been accomplished.

Altogether the records of 176 companies existing in 1907 were examined. Of these 90 have since been amalgamated into larger organisations, such as the Crown Mines, or the East Rand Proprietary Mines; 35 have been liquidated, some owing to failure, others because they have completed the mining of their ground, leaving only 51 in existence at the end of 1921. There are a few small companies, besides, that have made ventures, chiefly with outlying properties; but the amount of capital involved in them is so small as to make no perceptible difference to the result. The investigation may be regarded as covering the whole Rand during the period dealt with.

A preliminary point is the amount of debentures and loans issued by the companies. Financing by debentures is quite a subsidiary matter with the Witwatersrand mines, less than a tenth of the capital being supplied in this form. If a debenture issue is made and the obligations of interest and repayment duly carried out, the whole matter may be disregarded, so far as the present investigation is concerned; that is, it may be regarded as merely part of the working expenditure of the mine. But if default is made of a loan or debenture issue, that may reasonably be considered as offsetting the profits made in other parts of the whole industry.

Loans are made, mostly by finance companies which are interested in particular mines, and which are able to protect themselves, so that the whole amount lost is the comparatively trifling sum of £247,561.

There have been three cases of debentures defaulted on, the loss being some £600,000 nominal. These amounts we may suppose the imaginary investor to make himself responsible for, so they will be added to his outgoings.

There have been numerous reconstructions involving exchange of old shares for new, but these do not affect the result—only cash transactions involved in reconstruction need be entered in the account.

The average yield is, then, such a rate of interest that all the outgoings, accumulated at that rate, to a given date, will balance the income, similarly accumulated.* A rate is therefore chosen arbitrarily, and all the sums, in and out, are multiplied by factors that will represent accumulated interest to the date chosen, 31st December, 1921 (it makes no difference what date is chosen). Six per cent. was actually tried, and it was then found that the income appeared to exceed the outgoing. On recalculating with seven per cent. interest the position was reversed, so the true rate lies between the two, and was found by proportion to be 6.20%.

The actual calculation was as follows:—

	1000£
At 6% Loss on loans and debentures	1294
Other outgoings	226194
	227488
Income	230791
Excess	3303
At 7% Loss on loans and debentures	1375
Other outgoings	257221
	258596
Income	245298
Deficiency	13298
	3303
Proportion $\frac{3303}{3303 + 13298}$	= 0.20

which is to be added to the rate of 6%.

The yield of the mines is certainly not a high one, considering the erratic nature of the investment, some companies being very rich, and others failures; it is necessary to distribute investment over a considerable number of mines to feel any confidence in

* If the payment q ($+ve$ or $-ve$) be made at time t previous to the chosen date, the rate of interest i is given by the condition that $\sum qe^{it} = 0$.

the average return. Actually mining shares are usually dealt with on the Stock Exchanges on a basis of something like ten per cent. yield, and investments in real estate and industrial companies in South Africa probably bring in nearly as much. Over the question of leasing Government land in the Far East Rand for gold mining the Government Mining Engineer said: "On the financial side it will be assumed . . . that by a 'workable mining proposition' is meant a mine returning annually over a period of twenty years, after paying its share of profits to the Government, a dividend of at least 15 per cent. on the capital sunk."*

The result of the present investigation may be compared with certain statements published by the Mining Industry Commission of 1908.† That body made a serious attempt to estimate the profits that had been made by the Witwatersrand; its method was given as follows:—

"In order to ascertain whether dividends have been sufficient to provide amortisation and a satisfactory rate of interest upon working capital, the following course was adopted in the case of each company:—

The several subscriptions of working capital and the year in which each subscription took place were set down.

The annual payment required for the amortisation of each subscription of capital was calculated. These payments were assumed to accumulate at 3 per cent. compound interest.

The nett rate of interest which would be considered satisfactory to the investor was taken to be 7 per cent. (the Chamber of Mines quoted 6 per cent., Mr. Reyersbach 8 per cent. or 9 per cent.), and a calculation was made to arrive at the amount of accumulated interest due on each subscription of capital at the end of each year on this basis.

The first charges upon each dividend as paid were assumed to be:—

- (a) The payment to the sinking fund of such sums as were required to make it up to the amount at which it should have stood at that date had the calculated annual payments into it been regularly made from the dates

of subscription of the working capital and had accumulated at 3 per cent. per annum compound interest.

- (b) The payment to the subscriber of 7 per cent. interest for the current year and such arrears of accumulated interest as might be due with interest thereon at the same rate."

The result of this method of computation was to show a surplus of £7,116,000 over what was needed for amortisation and 7 per cent. interest. The total cash working capital from time to time subscribed for the establishment of the 64 producing mines in 1906 was £29,235,000. This, apparently, does not include capital subscribed to work ground that did not prove successful and was not absorbed into one of these 64 mines. It is not easy to compare their result with that worked out in the present paper, but it is highly probable that the return obtained from the early working of the mines was greater than during recent years.

It will be noticed that the Commission of 1907-8 suggested 7 per cent. as a reasonable return to the investor. There is considerable divergence of opinion on this point. The material prepared by the Commission seems not to have been preserved, and it is doubtful whether an investigator now would have access to the same sources.

That the profits were larger in the early days of the Rand than now is rendered likely, not only because the early exploiters of the field had at their disposition large masses of rich ore near the surface, but because they were producing a more valuable article. 1837-1907 was a time of very low prices, i.e., of a high value of gold; in the years 1907-14 gold was worth, on the average, quite ten per cent. less, in purchasing power, and since the outbreak of war it has, of course, fallen much more. Mining has become more economical, both on account of extension of scale and of technical improvement; but this has not sufficed to compensate the unfavourable situation. The technical advances have enabled the industry to survive—if primitive methods were still in use very few patches of the Rand would be rich enough to pay their way; but despite the improvement in technique the average interest now yielded by investment in the mines is not very much higher than that paid by Governments and other first-class borrowers.

* Memorandum on the Far East Rand (U.G. 20—1916), p. 7.

† Mining Industry Commission 1907-8 (Transvaal) Report, § 92.

SYMPOSIUM: MINERS' PHTHISIS.

Mr. J. Hayward Johnson: Dr. Orenstein in opening the Symposium asked for any small contribution that might suggest a line of investigation into this terrible occupational disease.

In September last, owing to the presence, presumably, of gelatinous silica, great difficulty was experienced in filtering an acid solution through a filter press (gold-bearing zinc from the precipitation boxes of a cyanide plant having been dissolved in sulphuric acid), which, though apparently clear, contained gelatinous matter. This gelatinous matter deposited on the filter papers forming an impervious film through which neither water or air would pass even under great pressure, within ten minutes a pressure of 60lbs. per square inch was registered. The above occurrence suggested the following questions to the writer:—

Is it not possible that a similar action takes place in the lungs of a person who inhales the fine, almost invisible, dust?

Is not this dust converted into gelatinous silica by being inhaled with gases that act upon the lung fluid as suggested by Mr. Marquard?

(Mr. Toombs' remarks, page 192, No. 10, Vol. XXI., regarding the formation of gelatinous silica in the mine sumps may have some bearing on this subject.)

Do dumpsmen or people living in the vicinity of dumps contract miners' phthisis?

In what form is silica found in the lungs, as minute grains countable on a konimeter slide or in a gelatinous form?

Is Potter's asthma or phthisis, which is said to be emphysema of the lungs, the same as miners' phthisis? I believe this disease is caused by the inhalation of the dust arising from the cleaning of the pottery prior to firing.

It is generally accepted that coarse grains of sand do no harm as they are more easily removed by expectoration. It would be thought that these larger grains would lacerate the lung tissue, which wound would, on healing, leave the "scar tissue" spoken of, and therefore be more harmful than the finer dust, unless the latter is converted as suggested above.

BOOK REVIEW.

REPORT OF THE TIN AND TUNGSTEN RESEARCH BOARD, Department of Scientific and Industrial Research.

H.M. Stationery Office, Imperial House Kingsway, London, W.C.2. Price, 3/6 (nett).

This report summarises the work of the Tin and Tungsten Research Board, formed in January, 1918, to carry on with increased funds the work on the same subject organised in 1916 by the Institute of Mining and Metallurgy. The details of the various researches are given in the twelve appendices to the report.

Attention was given chiefly to new methods of extraction, particularly those of a chemical nature, as it was not thought likely that great improvements could be effected in the present methods of treatment reached after many years of experience. Nevertheless the most immediately valuable information is probably that resulting from the investigations of Professor S. J. Truscott and his colleagues on the losses incurred in concentration. These investigations were in part completed just before the formation of the Board, but were followed up by others of a confirmatory nature, viz., microscopic studies (Hutchin and Meade, Appendix 3) and regrinding tests (Frechville and others, Appendix 5). The result of this work according to the report is: (1) That the presence of finely divided cassiterite cannot be avoided in mill products whatever method of crushing be used. (2) That excessive comminution of the mineral in regrinding is not to be feared. (3) That regrinding is necessary in dealing with "chats" (mixed grains), although the result is that a further quantity of minute particles of cassiterite are set free.

"The further question whether finely divided tin can be saved in ore dressing is answered in the affirmative by Professor Truscott." (Truscott, "Slime Treatment on Cornish Frames." Trans. I.M.M., Vols. 27 and 28.)

Other ore dressing questions investigated by the Board were, on the effect of various defloculators (Truscott and Yates, Appendix 4), on the application of flotation (Low and Birnie); and on a new process (Hadfield, Appendix 6) for the separation of minerals with different dielectric constants.

On the chemical side the researches are divided into three groups: (1) Direct solution of cassiterite or wolfram which gave no promise of success. (2) Furnace operations to convert the cassiterite and wolfram to soluble compounds which could be leached

and re-precipitated. Several processes under this heading were examined such as the Janson Annable (heating with producer gas and solution of the reduced tin in ferric chloride); the Oxland process (heating with sodium carbonate and leaching the product with water for the removal of sodium tungstate). Details of these investigations are given either in the text of the report or in the appendices, but, generally speaking, none of the processes were very promising. (3) Volatilisation processes: The investigation by Dr. O. J. Stannard (Appendices 9A and 9B) on the volatilisation of tungsten as tungsten oxychloride by heating concentrate mixed with a reducing agent (carbon) in a current of chlorine gas, were sufficiently good to warrant application for joint patent rights to the Government and Dr. Stannard.

The volatilisation of tin as stannous chloride was investigated by the Chairman of the Board (Sir T. K. Rose) and Mr. Goodchild, also with rather encouraging results, although the estimated cost in 1921, about £3 per ton, were prohibitive for ordinary Cornish ores. It is thought that the process may be applicable to low grade concentrate, say ten per cent., in cases where the production of saleable concentrate involves heavy losses.

The collapse of the tin industry in Cornwall unfortunately brought the work of the Board to a premature close, so that conclusive information on many points is still lacking. As it is, a very useful contribution to the metallurgy of tin and tungsten has been made, which should be of special value in South Africa. The report on the several chemical processes of extraction which have been proposed, although not altogether encouraging, will direct other investigators into the more hopeful lines of research for which, as may be seen from the excellent review by Mr. F. H. Michell (Appendix I.) of the present-day methods of tin dressing in Cornwall, there is a wide field.

H.R.A.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

COLOUR REACTIONS FOR ORGANIC ACIDS.—In the course of investigations on evaporator incrustations on sugar refining plants Mr. Müller found it of importance to verify the presence

of oxalates. A reaction was discovered which renders the identification of oxalic acid easy and conclusive. In this reaction 100 gms. of incrustation are washed until free from sugar (this is confirmed by the reaction with a naphthol and sulphuric acid), the residue treated with 300 cc. of dilute sulphuric acid, and the liquid filtered, evaporated and made up to 50 cc. Ten cc. of this solution are gently evaporated almost to dryness in a test tube, a pinch of pure resorcinol added, and 2 cc. of sulphuric acid allowed to run down the wall of the tube. A green or blue ring is formed, in the presence of oxalic acid, which, after agitation, assumes a magnificent azure blue, this becoming violet after heating.

The writer believes that this reaction is characteristic of oxalic acid and its salts. The principal organic acids which may or may not occur in plants have been tested, and it is found that none of them give the blue colouration peculiar to oxalic acid.

It is of interest to note that hydrocyanic acid gives a pink ring after two minutes, changing to orange after agitation, and crimson after heating, while thiocyanic acid gives bright red, orange-yellow and brown colours under the same conditions. The colour reactions of fifteen acids are tabulated.—C. MÜLLER, *Chem. Trade Journ.*, Int. Sugar Journ., October, 1922, page 523; November 3, 1922, page 541. (R.A.C.)

SCALE-FORMATION AND CORROSION IN BOILERS.—The writer discusses the effect of the impurities commonly occurring in boiler feed-water, and describes the mechanism of the corrosive reactions which are responsible for the deterioration of plates and tubes. When water is heated in a boiler the dissolved gases are driven off with the steam and there remains in the water all of the originally present solid impurities. Some of these are precipitated as sludge or as adherent scale, while others remain in solution and become concentrated, giving rise to other boiler troubles. Reactions between various impurities may occur, giving rise to precipitates or to strongly corrosive compounds. Calcium carbonate, or carbonate of lime, gives rise to sludge which gradually becomes converted to a hard scale. It is one of the least dangerous scales, as it is porous and is comparatively easy to remove. Sulphate of lime forms a dense crystalline scale. Sulphate scales are, with the exception of silicates, the most destructive and most difficult to remove. Sulphate of lime scales may be produced in a boiler whose feed water contains none of this substance, by interaction of magnesium sulphate with other lime salts which may be present originally or added in purifiers. In cases of this kind where magnesium carbonate is also produced, a further decomposition may result in the production of carbon dioxide, which passes forward with the steam and sets up corrosion in superheaters, turbines, etc. Carbonate of magnesia gives rise to carbon dioxide and flocculent oxide of magnesia, which latter is apt to clog the steam spaces of turbines, etc., if any priming occurs. Sulphate of magnesia is chiefly dangerous on account of the reactions which take place between it and other salts. With sodium chloride the products are sodium sulphate and magnesium chloride, the latter being one of the most dangerous constituents of boiler waters. Excess

of sodium carbonate in boiler feed waters ultimately causes extensive corrosion of brass and gunmetal fittings. At high temperatures and pressures caustic soda is formed and oxygen evolved, possibly by the production first of sodium formate which eventually is converted to formose; many boiler waters give positive reactions to tests for sugars. Further reactions may produce lactic acid, etc., and acetone while sodium carbonate is regenerated, thus completing a continuously acting cycle. Sodium sulphate, nitrate and chloride are regarded as innocuous except for the reactions which take place between the latter and magnesium compounds. Calcium nitrate is present in many waters; it is corrosive and is one of the four salts to whose presence the rapid corrosion of plates and tubes is generally due. These salts are calcium nitrate and chloride, and magnesium nitrate and chloride, and they should be regarded as present even though analysis indicates only the potential salts. Silica is very sparingly soluble, but soluble silicates may be present in considerable quantities. The use of a poor quality lime together with soda in softening plants will often result in the formation of silicate scales, which are intensely hard and compact, and are very difficult to remove.—A. A. POLLITT, *The Chemical Age*, July 22, 1922, Vol. VII., No. 162, page 76. (R.A.C.)

SODIUM HYDROSULPHITE ("HYDROS").—A booklet describing methods of analysis of sodium hydrosulphite has been published by Brotherton & Co., Ltd., City Chambers, Leeds. The indigo carmine method is customarily employed in the companies' laboratories, and the booklet gives full details relating to the preparation of pure indigo, the preparation of standard indigo carmine solution, and full details of methods and apparatus employed. In the silver method, which is also described, about 0.4 gm. of "Hydros" is weighed out in a stoppered weighing bottle. A silver solution is prepared by dissolving one gram of silver nitrate in 10 cc. of distilled water, and adding 15 cc. of 20 per cent. ammonia. This solution is poured on to the solid hydrosulphite and the reacting substances well mixed with a glass rod. After about five minutes the precipitate of silver is filtered off, washed well with dilute ammonium nitrate, dried, ignited, and weighed as usual. The silver may also be dissolved and estimated volumetrically; 216 parts of silver are equivalent to 174 parts of pure anhydrous sodium hydrosulphite.—*The Chemical Age*, August 5, 1922, page 173. (R.A.C.)

METALLURGY.

ACETYLENE NOT A PRECIPITANT FOR CYANIDE SOLUTIONS.—In connection with some investigations being made by the Bureau of Mines on the precipitation of gold and silver from cyanide solutions, it was suggested that the effect of acetylene for this purpose be investigated.

The acetylene used in the tests was from commercial calcium carbide. Experiments resulted in more or less complete precipitation of silver, but no gold.

As silver acetylene, obtained by precipitating the silver from an ammoniacal solution of

silver nitrate by acetylene, is yellowish white, easily soluble in cyanide and highly explosive, and the precipitate obtained from the cyanide solutions was black, not soluble in cyanide and not explosive, it was decided to investigate the nature of the silver precipitate from the cyanide solutions. In preparing a large quantity of the precipitate it was found that after complete precipitation of the silver the solution contained soluble sulphides. This provided a clue and after appropriate tests the precipitation was proved to be silver sulphide.

The conclusion drawn was that a small amount of hydrogen sulphide was contained in the acetylene and that this was the active precipitating agent. Other tests were made, all of which verified the conclusion that acetylene, free from hydrogen sulphide, had no precipitating power on cyanide solutions of silver or gold. Further details are given in a recent report issued by the Bureau.—JOHN GROSS, *Journal Franklin Institute*, July, 1922, p. 115-6. (J.A.W.)

ZINC DUST AND ITS USES.—The principal chemical properties of zinc assured the metal of important special uses from the commencement of the chemical and metallurgical industries, and to-day new outlets for the metal in these fields are constantly being discovered. The applications with which this article is principally concerned are those which employ the zinc in the powder form, a physical state in which the very large number of metallic particles present a considerable active surface, and thus allow very high speeds of reaction being obtained.

Those chemical properties of zinc to which it owes its extended use in the powdered state are: (a) Reducing properties, which are utilised particularly in the dyestuff manufacturing industry where zinc dust is used; for example, in the reduction of nitro compounds to organic amines, and in the preparation of sodium hydrosulphite for the reduction of the vat colours; (b) The property of precipitating metals, such as copper, cadmium, lead, silver and gold from their solutions. Interesting applications of this property reside in the purification of zinc-sulphate solutions destined for the electrolytic extraction of metallic zinc or for the manufacture of lithopone, and in the extraction of gold and silver by precipitation from their cyanide solutions; (c) The property of combining with dry oxygen only at a high temperature, and of giving with moist air a basic carbonate or with sea water an oxychloride which protects the bulk of metal from further alteration. These properties have led to the following applications for zinc powder: Painting of iron objects; galvanisation by cementation (Sherard's method, which consists essentially in immersing iron objects in zinc dust at a temperature of 300° C.); cold or electrolytic galvanisation of cast-iron objects, metallisation, or deposition by projection of a layer of zinc on the surface of the metal to be protected.

The industries which use powdered zinc obtain the material from different sources, in accordance with the chemical role it has to play. One of these sources is the thermo-metallurgical zinc (ordinary and electrothermic process) in which the zinc dust is but a by-product, whilst others

have to make use of zinc powder obtained by special means.

COMPOSITION OF ZINC DUST.

The composition varies with the process of manufacture, and the nature of the zinc ore used as the raw material. In the following table are given the composition of eight dif-

ferent powders. The first five samples were obtained by the ordinary retort furnace metallurgical process, the sixth is a crude dust from the Scandinavian electrothermic process, the seventh is a refined thermoelectric product, and the last is a special zinc dust made in the electric furnace from a non-calcined blende.

CHEMICAL COMPOSITION OF VARIOUS ZINC DUSTS.

Constituents.	No. of Sample.							
	1	2	3	4	5	6	7	8
Zn Metal	79.1	85.2	88.7	86.9	87.7	46.1	76.7	31.2
ZnO	11.2	7.5	6.6	10.1	9.8	48.5	20.1	48.3
Pb	1.9	2.0	2.5	2.1	2.0	6.1	0.2	2.9
Ag	0.01	0.001	—	—	—	—	—	—
Cu	—	—	—	0.02	0.01	—	—	—
Cd	1.8	0.5	1.3	0.2	0.1	0.1	0.07	0.13
As	Trace	Trace	—	0.27	0.17	0.05	0.01	—
Sb	—	—	—	0.15	0.11	Trace	Trace	—
Fe	1.32	0.79	0.30	0.01	Trace	0.58	0.05	1.80
CaO	—	—	—	—	—	—	—	Nil.
MgO	—	—	—	—	—	—	—	0.59
AlO	—	—	—	—	—	—	—	0.97
S	—	—	—	Trace	Trace	0.94	0.00	5.95
O	4.10	3.11	0.50	—	—	—	—	4.31
SiO	Trace	Trace	—	0.22	0.06	—	—	3.58
Insoluble Carbon	—	—	—	—	—	3.28	2.32	—
Zinc Nitride	—	—	—	—	—	0.34	0.33	—
Water	—	—	—	—	—	0.37	0.24	—

An examination of this table, even superficial, shows that zinc dusts are far from being pure zinc and that they contain varying amounts of foreign materials, more or less undesirable from the point of view of the ultimate application of the material.

Up to recent years the presence of carbon in zinc dusts was explained solely by mechanical entrainment of the element volatilised from the charge or from the electrodes. Lemarchands (*Revue de la Metallurgie*, December, 1920) was the first to draw attention to the relation between the carbon in the dust and the carbon monoxide in the distilling gases, and to propound a chemical origin, depending upon the reduction of the carbon monoxide by the zinc, to the carbon. The main reaction is $\text{Zn} + \text{CO} = \text{ZnO} + \text{C}$, but this is modified by two other reactions $\text{ZnO} + \text{CO} = \text{Zn} + \text{CO}_2$ and $\text{CO}_2 + \text{C} = 2\text{CO}$. Lemarchands has emphasised the importance of his observations by showing that it is of the utmost interest to make a precise determination of the equilibrium of the reaction $\text{Zn} + \text{CO} = \text{ZnO} + \text{C}$, if the conditions for obtaining the zinc dust or the condensation of the vapours as liquid metal are to be determined.

The oxidation of the zinc vapours by air is the most general and widely-appreciated cause of the formation of the zinc oxide. Air exists in the retorts and condensation chambers, from the very commencement of the distillation, and cannot be avoided either in the ordinary furnaces or in the electric furnaces. Zinc nitride, the proportion of which in a zinc dust may vary from 0.16 to 0.42 per cent., and the formula of which is Zn^3N_2 is also formed from

the nitrogen of the air in the plant. When the retort charge is not absolutely dry, the oxidation of the zinc is accelerated, the water vapour admixed with the air facilitating the formation of zinc oxide.

PHYSICAL PROPERTIES OF ZINC DUST.

Lodin, as a result of detailed microscopic examination, has shown that the grains of zinc dust are metallic globules surrounded by crystals of zinc oxide. Sieving of the powders No. 6 and No. 7 (in the table) gave the following figures for their size.

Size.	Percentage in the powder.	
	No. 6.	No. 7.
1. Above 1 mm.	40.0	26.4
2. Between 1 and 0.6 mm. ...	9.2	12.3
3. Between 0.6 and 0.3 mm. ...	5.7	10.8
4. Between 0.3 and 0.13 mm. ...	13.3	19.7
5. Between 0.13 and 0.09 mm. ...	6.2	7.7
6. Below 0.9 mm.	25.4	23.0
Loss	0.2	0.1
	100.0	100.0

The powder falling in the third and fourth classes (0.6 to 0.13 mm.) showed as white and clear crystals which were separately examined. Some which were soluble in concentrated hydrochloric acid were found to be pure zinc, whilst others insoluble in concentrated hydrochloric acid showed on analysis SiO_2 61.75 per cent.; Al_2O_3 11.52 per cent.; and ZnO 27.42 per cent. Some of these latter grains were transparent, others were not, and it was not possible to examine them in greater detail. They were

mainly found in ordinary retort-made zinc dust, and it is not impossible that their origin may be due to the furnace linings. Lemarchands states that it is this pellicule of zinc oxide which prevents the separate droplets of metallic zinc from reuniting, and which makes possible the obtaining of zinc dust by direct condensation. This question of the physical and chemical constitution of zinc dust must continue to be the object of profound and methodic study, and must effectively support the study of the condensation of the vapours of the metal, for this condensation constitutes a question of the first importance for the perfection of the metallurgy of zinc. The specific gravities of powders No. 6 and No. 7 have been determined by immersion in turpentine. No. 6 has been shown to weigh 5.81 grammes per cc., and No. 7 to weigh 6.10 grammes per cc.

FUSION OF ZINC DUST.

Heated in air, zinc dust owing to the concentration of oxygen on the surface of the metallic particles, ignites before melting. Several attempts have been made to effect this fusion in the absence of air or under the influence of compression, but the industrial results have not been satisfactory enough to warrant the introduction of such methods into zinc metallurgy.—A. BILAZ, *The Chemical Trades Journal and Chem. Eng.*, originally in *L'Industrie Chimique*, August 18, 1922, page 193. (F.W.)

MINING.

PRESENT VALUE IN ITS RELATION TO ORE RESERVES, PLANT CAPACITY, AND GRADE OF ORE.—Present value, sometimes called present worth or actuarial value, means the value to-day of the future earnings of an enterprise. In other

words, the present value of a mining property represents a sum of money on which there could be paid from operating profits a determined rate of interest, say 8 per cent., and in addition an amount which, if invested at compound interest as received, would at the end of the life of the mine pay back the present value.

The mere calculation of the present value of a mine, based on estimates of ore reserves, profit per ton, and plant capacity, may lead to grave errors unless very careful and mature judgment is used in connection with such calculations, for there are many factors which, though they do not enter directly into the figuring, have a very great bearing. Among these factors are probable average selling price of the product during the life of the mine, the cost of plant or of added plant capacity, the effect of increased production on the market for the product, and physical difficulties, such as limited water supply, limited labour supply, etc., any one of which might make it inadvisable or impossible to have a plant capacity which according to present value figures might seem the best one.

Ore Reserves and Plant Capacity.—Of two properties having the same grade of ore and the same operating conditions, the one having the greater ore reserves is not always the one having the greater value, for the reason that plant capacity enters into the calculations.

After a certain point has been reached, added ore reserves may mean little or practically nothing so far as present value is concerned, unless plant capacity can be increased.

The life of a property depends on its reserves of profitable ore and on the size of its treatment plant, and the larger its treatment plant and consequently the shorter its life, the greater will be its present value relative to the estimated total operating profits, assuming, of course, that the plant has been paid for.

TABLE 1.—Effect of Plant Capacity and Ore Reserves on Present Value.

Ore Reserves, Tons	Plant Capacity, Tons per Day	Life, Years	Profit per ton	Annual Profit	Total Operating Profit Over Life	Present Value	Per cent. of Total Operating Profit Represented by Present Value
26,000,000	5,000	15	\$2.00	\$3,500,000	\$52,000,000	\$27,700,000	52.3
52,000,000	5,000	30	2.00	3,500,000	104,000,000	36,700,000	35.3
104,000,000	5,000	60	2.00	3,500,000	208,000,000	42,200,000	20.3
104,000,000	10,000	30	2.00	7,000,000	208,000,000	73,400,000	35.3
208,000,000	10,000	60	2.00	7,000,000	416,000,000	84,400,000	20.3
208,000,000	20,000	30	2.00	14,000,000	416,000,000	146,800,000	35.3
208,000,000	40,000	15	2.00	28,000,000	416,000,000	221,600,000	52.3

Note.—All Present Value figures in this paper are based on 8 per cent. dividend, and on 5 per cent. interest on the amortization fund compounded annually.

No matter how much the ore reserves are increased, the annual profits will remain the same unless the size of plant is increased also, but with no increase of ore reserves annual profits will increase more than proportionately with increase of plant capacity—more than proportionately because costs will usually be lowered when greater tonnage is handled.

Grade of Ore Mined.—In studying the subject of ore reserves and their present value, one point which stands out as being very important, and which is not always given consideration commensurate with its importance, is the question as to whether or not the ore reserves can be divided into two or more sections or large blocks of different grades, and

if so, whether or not the higher grade portion or portions can be mined separately. Nearly always only the average grade of the total ore reserves is considered, although it might make a huge difference in the value of the property to the owners or shareholders if arrangements were made to mine the higher grade portion first, so as to pay them back their capital investment as quickly as possible.

Managers and superintendents often are quite satisfied if the grade of the ore mined keeps up to the average of the estimated reserves, and superintendents in order to make tonnage, and thus hold down costs, have at times been known to mix non-payable ore in considerable quantity with higher grade ore so long as the average was up to estimates, not realizing that such a procedure actually reduced the profits.

When the size of the plant of a property, for various reasons, is fixed or limited, the sending to the plant of ore of lower grade than

is absolutely necessary is a waste of plant capacity, and plant capacity costs money.

When it is possible, sometimes even at a considerably higher mining cost, to mine first and separately a higher grade portion of an orebody, it is nearly always advisable to do so, and in the case of very large low-grade mines it may be strongly advisable to do this, even if it means the eventual loss of large tonnages of ore on which a small operating profit could be made. Such mining might be called "selective mining," not in the ordinary sense of the term, but on a huge scale.

Let us assume: (1) that the ultimate practicable limit of plant capacity is 10,000 tons per day, or 3,500,000 tons per year, and (2) that it is 15,000 tons per day, or 5,250,000 tons per year. The figures in Tables 2 and 3 show the present value of the property if only the higher grade portion of the reserves is considered, and the present value of the total reserves if the higher and lower grade portions are mined together.

TABLE 2.—*Ultimate Limit of Plant Capacity 10,000 Tons per Day.*

	Tons	Profit per ton	Total Operat- ing Profit Over Life	Life, Years	Annual Operating Profit	Present Value
High grade portion ...	200,000,000	\$3.00	\$600,000,000	57	\$10,500,000	\$126,000,000
Total ore reserves ...	300,000,000	2.50	750,000,000	86	8,750,000	107,900,000

TABLE 3.—*Ultimate Limit of Plant Capacity 15,000 Tons per Day.*

	Tons	Profit per ton	Total Operat- ing Profit Over Life	Life, Years	Annual Operating Profit	Present Value
High grade portion ...	200,000,000	\$3.00	\$600,000,000	38	\$15,750,000	\$176,500,000
Total ore reserves ...	300,000,000	2.50	750,000,000	57	13,125,000	157,500,000

In connection with the question of grade of ore, superintendents and managers often dislike to leave low-grade ore behind, fearing that if they do it will never be recovered, but they fail to realize that to send low-grade material through the plant unnecessarily is wasteful, in that it uses up plant capacity, reduces annual profits and makes the enterprise as a whole less valuable to its owners. It is usually easier to mine the average grade than to arrange to mine the higher grade ore first, and sometimes the excuse is offered that the mining cost per ton of the latter would be higher. Such added cost, if it existed, would be in most cases, however, much more than balanced by increase in present value and by greater annual profits.

Ore reserves, whenever possible, should be mined in a way to give the property the greatest present value.

Losses of Ore in Mining.—In all cases of large operations there is a certain loss of ore

in mining, which may amount to less than 5 per cent., or may amount to 15 or 20 per cent. To determine definitely the amount of such loss is very difficult, for the reason that it cannot finally be proved until an orebody, or a large portion of it, has been mined out.

In any method of mining that contemplates the caving of the capping, or of both the ore and the capping, losses of ore are usually due to the admixture of ore and waste, the waste, representing either the overburden or the material along the sides of the orebody, or both. Generally speaking, the ore is drawn from the stopes or caved pillars, until the admixture with waste is so great that the grade of the mixture is not profitable. This means that some ore must be left behind. Such losses may or may not seriously affect the present value of a property, depending on its life, on the grade of its ore, and on the cost of mining.

In the case of properties having very low-grade orebodies, the present value is not

greatly affected by loss in ore and may not be reduced by such losses, provided the mining method employed is cheaper than would be the case with a method that would recover a greater percentage of the total ore; in the case of a property with a short life, the effect of ore losses on present value becomes more serious, even though the grade of ore is the same; and the greatest effect on present value is in the case of a mine having high-grade ore and a short life.

Except when the grade of ore is quite high, reasonable ore losses do not greatly lower the present value and are not nearly as important as the question of plant capacity or the grade of ore mined. The effect of ore losses on present value is due to the shortening of the life of a property; they do not affect the annual profits unless accompanied by a dilution of the ore and a consequent lowering of grade.

Time for Construction of Plant.—It is of great importance to consider the effect on the present value of the time consumed in the construction of the plant to treat the ore from a mine; in other words, the effect of the deferring of dividends, due to time required for plant construction. A simple example probably would be the easiest way to illustrate this point.

Let us assume the case of a mine that has 105,000,000 tons of ore developed on which an average operating profit of \$2 per ton could be made. Necessary funds have been raised for a plant with a capacity of 10,000 tons per day. The present value of the enterprise under different assumptions as to the length of time required for plant construction would be as follows:—

Time for Plant Construction.	Present Value.
Completed at present time	\$73,800,000
2 years	63,200,000
3 years	58,600,000
4 years	54,200,000
6 years	46,500,000

Often much time is lost in the consideration of processes or of plant design, and while it is wise to try to make an installation as perfect as possible, any loss of time due to what might be called over-cautiousness, or perhaps too great a fear of making mistakes, might be very costly.

The reason for these differences in present value is that the dividends which might have been received during the added time of construction would be deferred until the end of the life of the property; and in the case of properties having very long lives, such deferment would, for all practical purposes, mean a real loss, equivalent to the amount of the dividend.

Sorting.—In some cases, sorting of the ore, either underground or on the surface, may increase materially the present value of a property, especially when waste or extremely low-grade material can be sorted out and thrown away.

If sorting could be done without adding greatly to the operating costs, the output of a mill or smelter might be increased very considerably, which would mean a saving in plant capacity and a probable increase in annual profits, in spite of higher costs per ton.

Problems in connection with sorting are

often very complicated and should be looked at not only from the point of view of what is best for the mine operation, but also of what is best for the transportation and reduction plant operations. Each case is a distinct problem, and a careful study of any given case is often well worth while and many show that sorting would have an astonishing effect on the present value of a property.

As a simple illustration, let us take the case of a property under the following assumptions:

Total ore reserves	3,000,000 tons
Plant capacity	150,000 tons p.a.
Life of property	20 years
Gross value per ton	\$10.00
Total operating cost per ton	\$5.00
Profit per ton (without sorting)	\$5.00

The present value of the property, without sorting, would be \$6,800,000.

Now let us assume that the ore is of such character as to allow the sorting out of 25 per cent., with a loss of 4 per cent. of the gross value, and that the total operating cost per ton of sorted ore would be \$6.30. We would then have the following:

Total ore reserves (sorted ore)	2,500,000 tons
Plant capacity	150,000 tons p.a.
Life of property	15 years
Gross value per ton (sorted ore)	\$12.80
Total operating cost per ton (sorted ore)	\$6.30
Profit per ton (sorted ore)	\$6.50
Profit per ton (original unsorted ore)	\$4.87

The present value of the property, assuming that the ore was sorted, would be \$7,700,000.

In the above example, the advantage of sorting is not in the operating profit per ton of original ore, which is really lowered by sorting, but is in the reduced life of the property, which in turn increases the present value.

Purchase of Outside or Adjoining Properties.

—Mining companies, through fear of competitors or of undesirable neighbours, or fear that the price may be increased if they wait, often purchase properties adjoining their own, or isolated from it, even though the mining of ore from such properties may not be advisable or possible for 10, 20 or 30 years. Such purchases sometimes prove to be much more costly than might at first be supposed, for the reason that as long as they are not utilized they are what might be called "frozen assets," and the interest which might have been earned on the price paid for them is lost.

To pay a price of, say, \$1,000,000 to-day would be the same as paying about \$1,800,000 ten years hence; about \$3,200,000 twenty years hence; or \$5,700,000 thirty years hence, assuming an interest rate of 6 per cent.—E. S. BERRY, *Minina and Metallurgy*, July, 1922, pp. 11 to 16. (C.J.G.)

WASTE COAL IN THE TRANSVAAL.—The annual output of coal in the Transvaal amounts to upwards of 6,000,000 tons. The coal is usually of a brittle nature, so that a large amount of small coal, estimated at from 800,000 to 1,000,000 tons per annum, is produced in its mining and handling. A large proportion of this small coal is utilised

on the Rand, but about 25 per cent. is discarded on the dumps as waste. The accumulation of waste coal on the dumps in the Transvaal must now amount to a considerable total; during the war the question of the utilisation of this dump coal was investigated by the Imperial Institute. In the report furnished by the Institute to the Government of the Union of South Africa in January, 1918, it was indicated that the three principal means by which this waste material may be utilised were:—(1) Direct burning for the production of electric power; (2) briquetting; and (3) carbonisation for the production of coke and by-products.

These suggestions were considered by a Committee appointed by the Advisory Board of Industry and Science in South Africa to investigate the possibilities of utilising the waste coal. A further possible method for utilising the waste coal is that of pulverisation for use as fuel for locomotives, etc. The following account summarises the various proposals made as to suitable uses for the coal at present wasted.

A considerable amount of duff or fine coal is already used for direct burning in the Transvaal. In 1916, for example, upwards of 500,000 tons were burnt at power stations on the Rand, the Victoria Falls and Transvaal Power Company, Limited, being the principal consumers and accounting for 400,000 tons. The South African Committee consider that it is possible that further developments in this direction will help to decrease materially the amount of small coal now thrown on the dump and regarded as waste, but that this will depend on a greatly increased demand for cheap electrical power.

If the dump coal could be successfully briquetted it would be possible to utilise it for most of the purposes for which the lump coal is now employed. It is understood, however, that hitherto the question of a suitable binder has been an obstacle to any development in this direction in the Transvaal, and there is also some doubt as to whether the briquettes could be produced at a price to compete with lump coal, which realises a low price, on the average about 5s to 5s. 6d. per ton at the pit's mouth. The cheapness of lump coal in the Transvaal must have an important bearing on the question of the profitable utilisation of the waste. The South African Committee state that about 6 per cent. of pitch would be required as a binder in making briquettes with Transvaal fine coal, and that, if imported pitch had to be used, the cost of the briquettes would be prohibitive in view of the low cost of coal in South Africa.

The possibility of successfully using dump coal for coke manufacture will depend on the character and value of the coke obtained and the yield of the by-products, as well as on the demand for the latter in South Africa. It seems probable that coke of satisfactory character could be made from certain kinds of the dump coal, especially if it were briquetted or compressed before treatment, and that there would be markets in South Africa for sulphate of ammonia for manurial purposes and also for the benzol and tar.

In Natal several collieries produce coke of good quality by retorting a certain proportion

of the coal mainly in beehive ovens. Natal Ammonium, Limited, formed in 1913, uses anthracite coal of the Vryheid district, containing about 2 per cent. of nitrogen, for the production of ammonium sulphate. For this purpose the coal is screened to nut size. A certain proportion of the gas, resulting from the process, is carried into furnaces, and burnt under water boilers to provide steam power for fans, electric light, machine-shop and mining purposes; but the bulk of the gas, after a small amount of tar has been extracted, is wasted. The South Africa Carbide and By-Products Company was formed in the autumn of 1919 for the purpose of manufacturing calcium carbide and of extracting motor spirit, tar oil, and other by-products from the coal and shale of the Ballengeich Collieries. There is a contract for the supply of 1,000 tons per week of fines, small coal and pickings for the above purpose.

In the Transvaal, the Tweefontein United Collieries have an output of 1,000,000 tons of coal per annum. On the property is an enormous tonnage of inferior coal, containing on an average about 1.5 per cent. of nitrogen. For the treatment of this coal the Lymn-Rambush process of coal distillation has been adopted for the extraction of the nitrogen contents of the coal as ammonium sulphate. As much as possible of the gas produced in this process will be sold to the railways and other power users. The chairman of the Company anticipates that in time a production of 1,000,000 tons per annum will be treated for by-products only.—*Chem. Trade Jour and Chem., Eng.; The Iron and Coal Trades Review*, September 1, 1922, p. 297. (J.A.W.)

NATAL COAL MINING INDUSTRY.—The annual report of the Durban Chamber of Commerce states that the output of Natal collieries in 1921 totalled 3,141,129 tons, as compared with 2,965,699 tons in 1920. An improvement in the railway rolling stock placed at the disposal of the collieries was felt, a result of the more complete opening up of the Booth Junction—Cato Ridge line. The new line is carrying at present 12,000 tons per day, and considerably relieves the congestion in the lower part of the Natal main line. The shipment trade of the Province during the year was a record, due principally to the tonnage exported as cargo, which totalled 886,759 tons. There was a falling-off in the bunker trade as compared with previous years. The following are statistics regarding coal shipped at Durban within recent years:—

	Bunkers.	Cargo.	Total.
	Tons.	Tons.	Tons.
1921	944,163	886,759	1,830,927
1920	1,001,534	583,209	1,584,743
1918	592,283	583,167	1,175,450
1916	1,356,412	168,949	1,525,361
1914	1,048,033	375,663	1,423,746
1912	1,080,006	558,652	1,638,658
1910	1,246,115	323,344	1,569,459

Economic disturbances in other parts of the world are quickly reflected in the coal trade of Durban. The effect of the British coal strike, which lasted from April to July of 1921, was to reduce the volume of bunker trade, and to increase export. The suspension of the supply of Welsh coal to overseas markets caused

a demand there for South African coals. Large tonnages were exported from Natal to Indian ports and to Egypt, and in one or two instances cargoes were sent to South America; but with the settlement of the home dispute came a determined effort on the part of the Welsh collieries to recapture the markets which had been temporarily lost, and with their advantage in quality of coal and freight facilities they have succeeded in ousting competitors almost completely.

Coal for use at Capetown is still being carried by the overland route, a distance of 1,100 miles. The railage per ton from Hatting Spruit to Capetown (for bunker and cargo coal) has been reduced from 0.279d. to 0.242d. The railage per ton per mile is now 0.218d. The rate for bunker coal from Hatting Spruit to Durban is 28d. per ton-mile. In addition to the reduction in railage, the Government charge for loading at Capetown has been reduced from 3s. to 1s. 6d. per ton, which is 6d. below the pre-war level, while the charges at Durban remain, both for export and bunker coal, 50 per cent. higher than the pre-war level. The new Capetown railage rate is only 46 per cent. above pre-war, while the Durban rate remains at 166 per cent. above pre-war. It is believed that by extending the present export rate to apply to coal for Cape ports, Natal coal could be profitably carried by the sea route, and thus obviate the wasteful, long overland haul.—*Iron and Coal Trades Review*, September 1, 1922, p. 300. (J.A.W.)

MISCELLANEOUS.

PROPER USE OF COAL.—Bulletin No. 9, just issued by the Department of Internal Affairs of Pennsylvania, contains a paper by G. H. Ashley, State Geologist, advocating strongly a more economical use of coal, especially the coking-coals of the State.

The high volatile coals of Pennsylvania can be made to yield from four to six times the quantity of oil now being obtained from the State wells. In view of this, we are surely not justified in burning raw soft coal for raising steam when the yield of hydrocarbons is thus diminished. One authority, perhaps a little enthusiastic, has declared that within a few years it will be against the law to burn such coal. This is a desirable result, but it may not be very near at hand. Pennsylvania coking-coals were used for over eighty years in beehive ovens, but all the gaseous products were lost. It is not merely the tar and volatile hydrocarbons that are thus destroyed; a considerable amount of ammonia is wasted. The value of ammonia in modern industries is now recognized, especially when we recall the vast amount of money and scientific investigation Germany lavished on the synthetic ammonia process. Our ordinary methods of steam-making are wasteful of fuel as such, apart from the attendant objection from smoke. Greater efficiency has been obtained of late years, but even with the best conditions the results are not comparable to those obtained with liquid fuels in internal combustion engines.

The increasing use of gasoline and the rapid depletion of the oil reserves of the United States and Mexico will compel searches for other sources of liquid fuel. Two such sources are now attracting much attention, the oil-

shales and coal. Exploitation of the former is still in the experimental stage, but the distillation of coal is, in the main, a solved problem. By-product coke ovens are often operated at a rather high temperature. A slightly larger yield of coke can be obtained from the by-product oven than from the beehive form. Coke is a smokeless fuel, very much of the type of anthracite, but not nearly so compact and, therefore, burning more rapidly. The use of coke in domestic heating in the winter would be satisfactory along all lines except that a greater storage capacity would be needed and visits to the furnace would be somewhat more frequent. The proportion of light and heavy oil and of gas obtained from a given coal can be modified by the temperature, and experiments on the low temperature distillation are reaching the stage of commercial production in England, Germany and the United States. One ton of Kentucky coal containing 3 per cent. of moisture, 35 per cent. volatile matter, 55 per cent. fixed carbon and 7 per cent. ash, is said to yield 1,400 pounds of carbocoal, 30 gallons of tar, 9,000 cubic feet of gas, 20 pounds of ammonium sulphate, and 2 gallons of light oil thrown down from the gas.

The term "carbocoal" refers to a product obtained in Virginia by a special process of distillation. It is softer than ordinary coke and contains from 8 to 10 per cent. of volatile matter. It could be used at the point of production, but is said to be not suited to the general market. It is, however, employed in the manufacture of briquets, which are stated to be of good quality.

The tar contains no naphthalene but is rich in tar-acids and cresols, and contains more phenol than in other forms of tar. So far as Pennsylvania coal is concerned, it has been estimated from the results obtained in England, that by carbonizing the 140,000,000 tons of bituminous coal now mined in Pennsylvania, in a year, more than 400,000,000 gallons of motor fuel and a billion and a quarter gallons of Diesel and lubricating oils could be obtained. The total of these figures is about five times the quantity of oil now being produced in Pennsylvania. In addition over a million tons of ammonium sulphate would be obtained and the fixed residue, amounting to over a hundred million tons, would be an excellent smokeless fuel.—G. H. ASHLEY, *Journal Franklin Inst.*, July, 1922, p. 111-2. (J.A.W.)

BRIGHTNESS OF LIGHT SOURCES.—The brightness of several different light sources has been measured by different methods and previously reported. The values given in the table include some additional data, and some result previously published are added for purposes of comparison.—

Source	Brightness Candles per cm ²
Kerosene flame	1.3
4-watt carbon filament	51.0
0.9-watt tungsten filament... ..	227.0
Same lamp frosted	2.5
50-watt white mazda—outside	1.29
50-watt white mazda filament	408.0
2000-watt mazda—outside coil	864.0
2000-watt white mazda—inside coil... ..	2021.0
Same type lamp frosted—outside	105.0

—W. E. FORSYTHE, *Journal of the Franklin Institute*, February, 1922, Vol. 193, p. 252. (J.A.W.)

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA

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Proceedings

AT

**Special General Meeting,
10th February, 1923.**

A Special General Meeting of the Society was held in the Assembly Hall, Scientific and Technical Club, Johannesburg, on Saturday, the 10th February, 1923, at 8 p.m., the occasion being the unveiling of the War Memorial by Sir Lionel Phillips, Bart., to those members of the Society who fell in the Great War. Prof. G. A. Watermeyer (President) presided, and was accompanied on the platform by Major Forsyth Allen, Mayor of Johannesburg. There were also present:—

28 Members: Sir Lionel Phillips, Bart. (Honorary Vice-President), Messrs. H. R. Adam, C. J. Gray, F. W. Watson, J. R. Thurlow, J. Chilton, R. A. Cooper, J. Hayward Johnson, Dr. A. J. Orenstein, John Watson, H. A. White, J. A. Woodburn (Members of Council), Sir George Albu, J. A. Boyd, T. F. Considine, Andrew F. Crosse, J. M. Dixon, Jas. Gray, E. Homersham, W. Martin, Lt.-Col. E. Pam, R. C. Roe, G. Hildick Smith, S. H. Steels, F. Wartenweiler, H. R. S. Wilkes, Prof. J. A. Wilkinson, and A. A. Wood.

3 Associates: Messrs. H. D. Barnes, G. B. Gordon, and Samuel Evans.

About 200 visitors, including ladies, and H. A. G. Jeffreys (Secretary).

The President said the occasion for this Special Meeting of their Society was to show honour to those brave men, their former colleagues, who had fallen during the Great War. The idea of this was mooted by his predecessor in office, Mr. Jas. Gray, about three years ago. At that time they had no suitable place wherein to house the

Memorial, so the matter was dropped until they were in a position to find a suitable place. On the consummation of the Associated Scientific and Technical Societies they were able to find a very appropriate spot in which to house it. After they had got into that building, Prof. Wilkinson brought the matter up again and it was decided to proceed therewith. Their thanks were due to Professor G. E. Pearse for the design of the Tablet and arranging for the execution thereof. He now begged to introduce to them Sir Lionel Phillips, an Honorary Vice-President of 29 years' standing—he did not need much introduction; he would now call upon Sir Lionel to perform the ceremony of unveiling the Memorial.

Sir Lionel Phillips: Mr. President, ladies and gentlemen, the task I have undertaken to perform to-night is, in the first place, a very sad one, because it reminds us of friends and colleagues who have lost their lives; in the second place, it is a proud one, because we are doing honour to men who gave their lives in the great cause in which they believed. Many of them were friends of my own, and one misses them very much. One envies them, too, at the same time. I know that from this Society, which at that time numbered something like 920 members, they sent 148 of their number to the War. It is a very fine percentage; 16 per cent. of the whole; and, alas! 19 of them lost their lives. Moreover, a Society of this kind, containing as it does men of high qualifications, will necessarily always send a very large percentage of its numbers anywhere where good work is to be done. To my knowledge, many more men belonging to the Society would have volunteered and gone away from here, but they felt it their duty—in fact, it was impressed upon them, or almost insisted upon in many cases, that it was their duty to remain at the helm and

produce what was almost as necessary to the war as the services of the men themselves, namely, gold. On that ground, they remained at the post of duty; otherwise the numbers who would have gone from here to serve their country would have been even greater than they were.

Ladies and gentlemen, whenever war occurs anywhere there are always large numbers of men ready to volunteer. It is rather amazing in one way; it is a tribute to human kind on the other, because, unless we had ambitions and a great sense of what we owe to the country to which we belong, it would be a sad day for patriotism. But, strangely enough, war has always been pictured, from time immemorial, in very attractive terms; terms that make the younger men particularly long to take part, with the idea of winning glory and also the vision that going into a gallant struggle is something which they very likely will enjoy. Men who have been in war and seen the horrors of it in the sordid time of war come out afterwards with a very different impression of war happily, and, like some other dreadful things in life, the impressions of all the horrors that men go through in time of war do not seem to be stamped so deeply upon their minds as to prevent them, perhaps happily, from volunteering for service when the occasion arises again. So to the end of time, I suppose, ladies and gentlemen, men will be found—and it is a sound condition of things in humanity—who are ready to take their places in the service of their King and country when the circumstances need it.

Now, ladies and gentlemen, it is a more remarkable thing, when we think what a devastating thing war is. Looking at this terrible war we went through, which has cost the lives of so many millions of people, what is the result? The whole world almost is lying prostrate, and we do not even now know when we are going to get back to conditions which can be reckoned in any sense normal. So that, war looked at from that point of view, is the most senseless thing that was created in the world. On the other hand, it has one very wonderful side. It certainly makes for the humanity of mankind, and certainly makes for the chivalry of a nation. Without war, I suppose—I do not know, but I cannot imagine what could take the place of war which would keep up a spirit of chivalry. Men will require to have, Heaven knows what, in the place of war; but it is one of those directions in which the best, the strongest, the most vigorous

and noble actions of men are found. So I suppose it may have been ordained that, as we have had it from time immemorial—from Biblical days it was ordained perhaps that humanity is intended to struggle, and, when their struggles cannot be settled by some argumentative combat, that they should then take to this other form, which is the most brutal and most horrible imaginable.

Ladies and gentlemen, one other word. I am not going to detain you at great length; but there is one other word I should like to say about the monument—not this special monument, but monuments generally. This monument, I know, has been designed with great taste, with great simplicity—as it should be—for the purpose for which it was intended, and to remain for all time as a record of some of our own associates who gave their lives.

Monuments generally, ladies and gentlemen, are, of course, in the first place, erected to do honour to the dead, and in that way record of the deeds are kept fresh in the memory. The acts of those men whose names are recorded; but, in addition to that, they also—and there, I think, there is much to be said in favour of monuments generally—stimulate the young idea to efforts of sacrifice and of valour and of courage when the circumstances arise; they inculcate into young men a spirit of heroism, without which a nation would be a very poor thing indeed.

As I said before, ladies and gentlemen, of the 19 men who gave their lives in the Great War, many of them were known to me; one in particular, I should say, I regarded as a friend, whose loss was, perhaps, without being invidious in mentioning any one name, the greatest loss these Fields suffered, that was Mr. Frederick Trump. He was a gallant man; he rose to be a Lieut.-Colonel, and was very much beloved by a great number of people who had the honour of knowing him. There were other men of equally good service no doubt, less familiarly known to some of us; equally gallant and splendid men who could ill be spared. And, ladies and gentlemen, I think I must not say any more, or I shall perhaps be making you more unhappy than I have made you already. But one cannot perform any function of this kind without feeling the origin is a deep and almost religious function; and, perhaps, you will forgive me if I have not attempted to do anything which should lighten the proceedings in any way.



I will, therefore, proceed now, with your permission, Mr. President, to perform my task of unveiling the Memorial.

Sir Lionel then released the Union Jack which had covered the Tablet, and two buglers detailed from the Imperial Light Horse sounded the "Last Post."

Prof. J. A. Wilkinson : Mr. President, ladies and gentlemen, the names of the men whom we commemorate this evening are:—

MAX STANFIELD EATON ARCHIBALD.
EDMUND GRAHAM BASKETT.
CHRISTOPHER YULE BRUCE.
THOMAS HENRY CLESHAM.
WILLIAM JAMES N. DUNNACHIE.
ARTHUR RICHARD FLYNN.
WILLIAM GODFREY FROUDE.
FREDERICK NIVEN GERDS.
JAMES VAUGHAN HARRIS.
CHARLES KENELM DIGBY JONES.
BERNARD ANGUS JOHNSON.
ALEXANDER DOWNING JOHNSON.
WILLIAM NICKLIN.
WILLIAM HENRY PICKBURN.
WILLIAM HARRY POYNTON.
REGINALD WILLIAM ROBINSON.
GEORGE SIMPSON.
DOUGLAS WILLIAM STACEY.
FREDERICK JOSEPH TRUMP.

The "Reveille" was then sounded, and the meeting terminated.

Lient. Max Stanfield Eaton Archibald, Lient., Royal Engineers, joined the Society in September, 1911, and at one time had been employed at the Knights Deep.

Lient. Edmund Graham Baskett, an Associate of the Society since January, 1911, was a native of New Zealand, and came to South Africa in 1910, holding various appointments at the New Modder, Modder B., City and Suburban, and Rietfontein T.C.L., and afterwards that of Reduction Officer at Prestea, Block A, West Coast of Africa. This position he resigned on obtaining a commission in the Nigerian Regiment, serving with the West African Frontier Force until nearly the end of the war. Influenza, together with pneumonia, caused his death on October 27th, 1918.

Lient. Christopher Yule Bruce, Royal Engineers, became a member of the Society in June, 1912, when mine captain at the Simmer Deep, Ltd. He was killed in action in France on 28th March, 1918.

Lient. Thomas Henry Clesham, Royal Engineers, was a member from April, 1911, and had been employed at the Simmer East, Ltd., as a sampler.

Lieut. William J. N. Dunachie, who joined the Society in March, 1907, had been associated with the Consolidated Langlaagte Mine, the Geldenhuis Deep (West Section), and the New Rietfontein Estate and G.M. Co., Ltd. Later he became the Manager of the Gartness Colliery, Airdrie. Soon after the war started he received a commission in the Royal Engineers, and went to France in charge of a section of the Tunnelling Corps. He was killed in action on the 24th April, 1915.

Lieut. Arthur Richard Flynn, Natal Carbineers, joined the Society in September, 1912. He was employed on the Knights Deep, Ltd., as reduction works officer.

Lieut. William Godfrey Froude, who joined the Society in April, 1912, had been employed on various mines on the Rand as a mine captain, and later in Rhodesia. Mr. Froude joined the Imperial Light Horse as a Lieutenant, and was killed in action on the 25th November, 1914, during the operations against Kemp in the Western Transvaal. He had previously distinguished himself during the South African War, on the Bechuanaland Border, being mentioned in despatches and receiving the D.C.M.

Lieut. Frederick Niven Gerds, A.R.S.M., St. Inst.M.M., joined the Society as a member in April, 1912, when practising as a mining engineer at the Cam and Motor Mine, Rhodesia. He left for England in April, 1915, was gazetted as a 2nd Lieut. in the 176th Tunnelling Company of the Royal Engineers shortly after his arrival, and died from gas poisoning on the 2nd June, 1915.

Capt. James Vaughan Harris became a member of the Society in July, 1905, and prior to the outbreak of hostilities was cyanide manager at the Alice Mine, Rhodesia. He gained the rank of Captain and was awarded the Military Cross, being killed in action in Palestine.

Capt. Charles Kenelm Digby Jones, a member since 1905, was at one time manager of the Jumbo Mine, Salisbury, Rhodesia. He died of pneumonia in Russia while on active service.

Lieut. Bernard Angus Johnson, 5th Battalion, the Buffs, joined the Society in June, 1912, and had been employed on the Simmer

Deep as a reduction worker. He was killed in action in Mesopotamia on the 21st January, 1916.

Lieut. Alexander Downing Johnson, 2nd South Staffordshire Regiment, had belonged to the Society since November, 1903. Prior to the war he was employed on the Jessie Mine in Rhodesia. He was killed in action on 25th September, 1915.

Lieut. William Nicklin, joined the Society in November, 1903. He served in the South African War with the West Somerset Yeomanry, and soon afterwards qualified as an assayer, finding employment in that capacity on the Witwatersrand. He joined up with the Royal Engineers early in the war and was gazetted 2nd Lieut. on 4th March, 1916, being killed in action on the 24th August of the same year.

Major William Henry Pickburn, South African Heavy Artillery, joined the Society in September, 1911. He was wounded in action at Le Touquet, France, on 13th November, 1916, and died the same day. He was mentioned in despatches.

Sergt. William Harry Poynton, had joined the Society in April, 1913, being then engaged in metallurgical work in Australia. He enlisted in 1915 in the 27th Battalion, 3rd Reinforcement, Australian Imperial Force, and lost his life in February of the following year.

Capt. Reginald William Robinson, 5th Royal Irish Fusiliers, was killed in action in the Dardanelles on 15th August, 1915. He was a mining engineer and engaged at one time at the Penhalonga Mine.

Lieut. George Simpson, metallurgist, became a member in August, 1910. He served on the Western front, obtaining a commission in the 18th Durham Light Infantry. He was killed in action in France on the 4th July, 1916.

Lieut. Douglas William Stacey had been connected with the Society from January, 1908. At the outbreak of the war he was engaged in mining in Rhodesia. He joined the Royal Flying Corps in 1916, gained his commission, and became a pilot. He died of wounds received in France in May, 1917.

Lieut.-Col. Frederick Joseph Trump was born in 1872, and was the second son of the

late Mr. H. V. Trump, J.P., General Manager of the Rhymney Iron Company. He started his mining career in the service of that company, attaining the position of colliery manager until in 1901 he took up an appointment as one of His Majesty's Inspectors of Mines, which position he held until 1911, when he left the old country to come to South Africa. After spending a few months in the Consulting Engineer's Department of the Corner House he was appointed assistant manager and later acting manager of the New Modderfontein Gold Mining Company, Ltd. On the 12th December, 1911, he was appointed manager of the Ferreira Deep, Ltd., which position he held up to the date of his death. He returned to England early in 1915, and was given back the rank of Captain in the 1st Mon. Territorial Regiment, which he held before leaving England. After serving some months in England he left for France in October, 1915, where he was promoted to the rank of Major, and subsequently transferred to the 16th South Staffordshire Regiment, with the rank of Lieut.-Colonel. Whilst serving in France he obtained the decorations of "D.S.O." and "Croix de Guerre." On the night of December 1st, 1917, he and his company were relieved in the trenches and retired to rest billets, but on the following morning about 8 a.m. he was killed whilst in his bed, and apparently whilst asleep, by a shell or bomb from a 'plane. He was buried the next day at Sailly la Bourse with the fullest military honours possible. The Brigadier-General and his staff attended the funeral.

Extract from letter from London Secretaries, Ferreira Deep, Ltd., 14th December, 1917, giving portion of a letter from Mr. W. H. Trump, of Rhymney:—

"The only particulars received to date are contained in a letter from Major Hutchence, 1/6th S. Staffordshire Regiment, who says that a shell or bomb dropped quite near the hut where my poor brother was sleeping, about 8 o'clock on Sunday morning last, and that he was killed instantaneously in his sleep.

"When home on leave last he particularly mentioned his wish that the whole wretched business would soon be over so that he could return to what he used to refer to as 'his mine.'"

Proceedings

AT

Ordinary General Meeting, 17th February, 1923.

The Ordinary General Meeting of the Society was held in the Assembly Hall, Scientific and Technical Club, 100, Fox Street, Johannesburg, on Saturday, the 17th February, 1923, at 8 p.m., Prof. G. A. Watermeyer (President) in the chair. There were also present:—

28 Members: Messrs. H. R. Adam, C. J. Gray, F. W. Watson, J. Chilton, Andrew King, H. Pirow, J. M. Thorburn, John Watson, A. Whitby, H. A. White, J. L. Willey, J. A. Woodburn (Members of Council), Dr. W. A. Caldecott, J. M. Dixon, J. Henderson, E. Homersham, Geo. Melville, P. T. Morrisby, J. V. Muller, Prof. H. H. Paine, Lt.-Col. E. Pam, T. K. Prentice, R. C. Roe, R. A. H. Flugge-de Smidt, S. de Smidt, J. T. Triggs, F. Wartenweiler, and H. R. S. Wilkes.

6 Associates: Messrs. H. D. Barnes, G. J. V. Clarence, O. A. Gerber, H. L. Krause, H. Rusden, and L. Ussher.

1 Visitor (Dr. R. A. Lehfeldt) and H. A. G. Jeffreys (Secretary).

MINUTES.

The President asked whether it was the wish of the meeting to confirm the Minutes of the last meeting as published in the *Journal* for November, 1922. This was agreed to.

NEW MEMBERS.

A ballot was taken for the election of new members, and the following gentlemen were declared unanimously elected:—

CHILDE, F. R., Principal, Government Miners' Training School: Wolluter School, P.O. Box 82, Jeppestown, Johannesburg.

FROST, P. O'B., P.O. Box 8, Messina, Transvaal: Mine Surveyor.

HOSKING, F. J., P.O. Box 2, Randfontein, Transvaal: Assayer.

LETCHER, I. OWEN, P.O. Box 963, Johannesburg: Mining Engineer.

MEYER, T., 4, Myer's Mansions, corner Bree and Von Brandis Streets, Johannesburg: Mining and Metallurgical Engineer.

MULLER, J. V., Robinson Deep, Ltd., P.O. Box 1392, Johannesburg: Mining Engineer.

MUNRO, G. R., 21, Soper Road, Berea, Johannesburg: Chemist.

OLVER, W. W., Randfontein Central G.M. Co., Ltd., P.O. Box 2, Randfontein: Assayer.

PARRY, C. S., Randfontein Central G.M. Co., Ltd., P.O. Box 46, Millsite, Transvaal: Chief Assayer. (Transfer from Associate Roll.)

PAULL, T. H. L., P.O. Box 98, Langlaagte, Johannesburg: Mining Engineer.

PERLSTEIN, A., Managing Director, Orenstein & Koppel, Ltd., P.O. Box 7395, Johannesburg.

ROE, R. C., Manager, The Gandy Belt Manufacturing Co., Ltd., P.O. Box 2017, Johannesburg.

SIDDALL, J. L., Joint Manager, Central Mining and Investment Corporation, Ltd., P.O. Box 149, Johannesburg.

The Secretary announced that the following gentlemen had been admitted by the Council as Associates:—

ASSOCIATES.

BARNES, H. D., S.A. Inst. for Medical Research, P.O. Box 1038, Johannesburg: Laboratory Assistant.

CARSON, G. M., 87, Bedford Avenue, Benoni, Transvaal: Foundryman.

DAUDE, E. F. B., Owyhee Hotel, Boise, Idaho, U.S.A.

HALL, A., P.O. Box 345, Johannesburg: Engineer.

USHER, L., "Thorganby," St. George's Road, Parktown, Johannesburg: Inventor.

GENERAL BUSINESS.

The President drew the attention of members to the fact that two of their number had passed the Mine Managers' Examination, namely, Messrs. C. W. B. Jeppe (Honours) and A. W. Runciman. (Applause.)

THE VALUE OF RAND GOLD MINING INVESTMENTS.

By W. A. CALDECOTT and
G. J. V. CLARENCE.

(Printed in *Journal*, December, 1922.)

The President said he would ask Dr. Caldecott to introduce his paper published in the Society's *Journal* during the recess.

Dr. W. A. Caldecott said that in the absence of Mr. Clarence, who was to have introduced the paper, he would merely refer to the fact that the Society had received at times valuable papers on economic subjects but of a somewhat general or abstract nature. In the present case one specific aspect was dealt with of what might be termed practical or applied economics, and the authors trust that the considerations set forth in the

paper may be of some service to members, and they will be pleased to elucidate any points raised in discussion which are not adequately dealt with in the paper.

The President remarked that members had probably read the paper in the *Journal*, and they must agree it was a very valuable paper. The authors had given curves which showed graphically the results of the various formulae used in the estimation of present values and the lives of mines. It was very essential to note that the curves were sufficiently accurate to represent the data at one's disposal. Meticulous accuracy in calculation did not always attain its end, because very frequently they might be perfectly correct in their calculation, but may have used the wrong formula. So those curves which Dr. Caldecott and Mr. Clarence had prepared showed at a glance the effect of varying the factors. For those reasons he thought they were very much indebted to the authors for their paper, and he would ask members present to accord them the usual vote of thanks.

He would next call on Dr. Lehfeldt to present his paper published in the *Journal* during the recess.

RETURN TO CAPITAL INVESTED IN THE WITWATERSRAND.

By R. A. LEHFELDT.

(Printed in *Journal*, January, 1923.)

Dr. R. A. Lehfeldt said he would not keep them more than a few minutes in talking about his paper which was already printed, the subject of which was, how much profit the Gold Mines of the Witwatersrand make? a subject which he hoped might be of some interest to Johannesburg, for, after all, it was what they all lived on.

It had been said that, in order to get a sovereign's worth of gold out of the ground people usually spent about 30/- to get it. That, he believed, was no exaggeration with regard to some mining fields. Of course the mining fields here were very different from the average mine field where they had scattered quartz veins. Here the deposits were so regular that gold mining had become an industry rather than a gamble; but even here he thought the profitableness of gold

mining was exaggerated in the public's estimation. It was impossible to get at the facts as thoroughly as one would like. The only way to form a real estimate of what these Fields here had been worth would be to go back to 1886 or 1885, or when they really began, and get an account of all the money that had been spent on developing mines here. Well, such records were not in existence to-day—or, at least, he did not think so. As a matter of fact, the Mining Industry Commission of 1907 did try to dig out as much information as it could, but he did not know that the records which they had got together were in existence now; he had not been able to discover them.

The Commission at that date formed an estimate of how much the mines had yielded, but he thought their investigations were really confined to the mines which had, at that time, succeeded. A lot of money was, of course, sunk in ventures which were unprofitable, and of which no record had been kept. The trouble was to know how to do that when they had numerous attempts at developing mines here and there, sometimes straightened out and abandoned, one syndicate amalgamated with another, and all that sort of thing going on. The problem would certainly be beyond his capacity to go back to the origin in that way. What he had tried to do was to take a period, say, from 1907 to the present time and make an estimate of the profits yielded. In order to choose fairly the period for investigation he had begun by getting a curve plotted of the market price of the Rand mines' shares, taking that as being the most typical, and trying to find a certain date when they were neither unduly depressed nor unduly elevated in price—that was to say, to try and get away from the effect of the booms and slumps which were so characteristic of gold mining and the markets.

The date chosen, or the start, was the 30th June, 1907, and he had brought the figures up to the 31st December, 1921. There was no special reason for choosing one date rather than another; so he had brought it up to as recently as he could.

The actual investigation was carried out for him by the officials of the Mines Department; Sir Robert Kotze was kind enough to place their services at his disposal, and without that assistance it certainly could not have been done, because of the enormous amount of work involved in digging out records, tabulating, arithmetic, and so on. He would like specially to mention Mr.

Vaughan—in charge of the Statistical Department—who had taken a very lively interest in the result.

The method was this. Supposing that an investor had, at the earlier of those two dates, bought up the Witwatersrand; supposing he had then proceeded to supply all the capital that was supplied afterwards for developing new mines, making extensions and so on, and had pocketed all the dividends, and, at the later of those two dates, had sold out his holdings at market prices; how would he stand? What sort of investment would it have been to him? He need not go into the arithmetical details—which they would find in the paper; he would endeavour to be as brief as he could, and so afford opportunity for a few questions afterwards, if it was not against the rules. Well, the result of that estimate was that the nett yield on the capital would amount to 6.2 per cent., that was to say, if that investor had put all his money into a security which yielded 6.2 per cent., then at the end of 1921 he would have been just as well off as if he had worked the Witwatersrand. 6.2 per cent. was more than they could get out of Government securities; they could get 5 per cent. or $5\frac{1}{2}$ per cent. now. At the earlier of those periods they could only get $3\frac{1}{2}$ per cent. or 4 per cent.; but, by judicious investment in sound securities, not gilt-edged, all through that time, they could have got something like $4\frac{1}{2}$ per cent. or 5 per cent. at the beginning, and 6 per cent. or $6\frac{1}{2}$ per cent. at the end of the time. Insurance companies, which were, naturally, very careful investors, made rates of some—think like that. Consequently, an imaginary investor taking over the risk of running the Witwatersrand did not get, as compensation, any great excess over what he might have got by putting his money into securities in the way an insurance company manager would do. That result was, to his mind, sufficiently striking. It showed that gold mining on the Witwatersrand during the last 15 years had not been at all a paying sort of investment. If they asked the average commercial man whether he was prepared to put £100,000 into a rather speculative business which was supposed to yield 6.2 per cent. it might yield more, or it might yield less, he would very likely turn the venture down.

The importance of getting accurate statistics in this matter was obviously great, because the Witwatersrand needed more capital. If more mines were going to be

developed it was necessary to know what sort of return was likely. Of course, money was raised as a matter of fact very commonly during the speculative period on the Stock Exchange, when people's expectations were a good deal higher than the truth; but in the long run the prosperity of the industry had to depend upon providing adequate remuneration for capital. Opinions had varied very much as to what would be a satisfactory return to tempt the capitalist. In the Report of the Mining Industry Commission in 1907 a rate of 7 per cent. was talked about; but the witnesses on behalf of the Chamber of Mines, he thought, hardly felt that it would be very satisfactory to offer a return which averaged only 7 per cent. In the Memorandum on the Far East Rand which Sir Robert Kotze published in about 1915 or 1916 he talked about developing mines in the Far East Rand, and formed an estimate as to what they ought to yield, and spoke of 15 per cent. as being satisfactory. Of course, now, if it was a question of developing a new mine on the Far East Rand naturally it would have at its disposal improvements in mining engineering, which had made very great progress since the Witwatersrand had started. So that it was not quite fair to compare it with the average yield which had been turned out up to the present.

With regard to the profitableness of mining on the Rand; as he had said, the period he had chosen, during the last 15 years, would be much more interesting if one could go back to the origin. If one were able to do that no doubt one would find a higher rate. In the early days of the Witwatersrand, there were numerous mines which apparently did very well; they struck good stuff to start with; and there was also an important economic reason why profits were better in those days—that was, the product was more valuable; gold was more sought after and was more valuable. Of course that brought up the old, little trouble about the valuation of gold as compared with other things. He did not want to waste their time in talking about economic principles, but he would just point out, seeing they were estimating the value of things in terms of gold, that they had to invert the process in order to estimate the value of gold in terms of other things.

During the early '90's and 1900's the price of things was low; that meant that gold was, relatively, valuable; prices were low; gold was falling in value up to the end of the

century. Since that time prices had risen violently and gold had become of considerably less value. Therefore, if they were looking to the future to see whether these figures were any guide as to the future, the most important point was, what sort of value they were likely to expect for gold. He was not going into that question that night; but he would draw their attention to the fact that it was on that the results depended.

In the first fifteen years or so of the Witwatersrand's existence, when gold was relatively valuable, it was comparatively easy to make a mine pay, and, although the mines were not so economically managed from the engineering point of view as they are now, the profits were apparently good.

During the period of 1906 to 1921, which he had been considering, the situation was not so favourable as regarded the value of the output, but the mines had benefited by considerable advances in mining engineering. The natural result of those two influences was that rate of profit.

With regard to the future the main inference was the continued improvement in the technique of mining, and, on the other hand, whether gold was going to be more or less valuable at the present time.

Mr. F. Wartenweiler, in moving a vote of thanks to the author for his paper, regretted that he had, as yet, not read the paper carefully. However, he understood from Dr. Lehfeldt that evening that the nett return on all the mines of the Witwatersrand over the period stated was at the rate of 6.2 per cent. That, of course, was a low figure for a mining enterprise. It might have its value in Governmental circles by showing how poor the return really was and indicate to them that the mines were overtaxed. If they worked out with the same amount of care the statistics from other groups of industrial enterprises, such as steel, cement, or any well established industries over a term of years, they might find the return was not better. There was always the speculative element and the personal element which exercised its faculty of discrimination between the good and the bad. He thought, before they completed the discussion on the paper, it would be only fair if Dr. Lehfeldt would show them the nett return on some of the better mines over a complete life. A mine like the Robinson, which had about completed its long life, would be a good example.

The President announced that Dr. Lehfeldt would be very pleased to answer any questions on the paper which might be asked by members present.

Mr. E. Homersham asked Dr. Lehfeldt whether he had made any allowance for the shares issued at a premium. Some companies have been able, at convenient times, to issue £1 shares at a premium of £3 to £4. A man who had bought shares at a premium and only got 6 per cent. on the nominal value would not be very satisfied.

Dr. Lehfeldt, in reply, said he had tried to explain what was the method adopted, and, if the gentleman would examine the point, he would see it gave an answer to the question. He had supposed his imaginary investor to buy up the Witwatersrand at market price on the 30th June, 1907, and found out what would happen to his investment after that date; that said nothing about what would be the return. But, as regarded an original investor in one of the earlier mines who bought shares at par, or at a premium, well, as he had said, he had not the material to go back further; but, starting at that date, with the market valuation on the Witwatersrand—and it was a date when the market was not unduly boomed or depressed—from that time up to nearly the present time, the return made on the capital then invested would be 6.2 per cent.

Mr. A. Whitby asked what effect reconstruction had on profits.

Dr. Lehfeldt said the effect of reconstruction was included in the result. Supposing, for instance, that Mr. Whitby had originally invested in the four or five mines which went to make up the East Rand Proprietary, his capital was rearranged at the time of the amalgamation; but that was included in the investigation, because he (the speaker) had supposed his investor to buy up all the shares of those half-dozen mines and to get, in return, dividends which were paid by the E.R.P.M. He would see the actual process of amalgamation was merely the exchange of certain pieces of paper for certain other pieces, which made no difference to the result. If, at the time of the amalgamation, or at any other time, some cash capital was subscribed, that had got to be allowed for in the investigation. He had

supposed his investor to put up that money and had debited him. So that amalgamation did not make any difference to the result.

The President said another point he would like to raise was, the variations which had taken place with regard to loss on return of capital, or gain on return of capital. He would like Dr. Lehfeldt to give members some idea as to the range of variation in his figures.

Dr. Lehfeldt replied he was sorry he could not do that then. He had got the material and he thought he could give them a supplementary note on that point, when he had worked it out. He would tell them the form in which the material existed at the present time in the case of each mine. The officials of the Mines Department who had done it for him worked out all the sums that were paid in the imaginary acquisition of shares, and also any shares or debenture capital subsequently subscribed, and also all the sums received in the form of shares. Consequently, they would see that a sort of balance sheet was made out for each mine separately. That figure of 6.2 was arrived at by equating the two sides of the balance sheet, taking the Rand as a whole. If they took any individual mine they could, with sufficient trouble, make out a corresponding balance sheet, and that would give them a rate which in some cases would be more and in some cases less than 6.2 per cent. He had not worked that out; it would involve an enormous amount of arithmetic: but what he could do was this. Take the figures which he had got at 6 per cent. and say that for each particular mine the profit, or loss, was so much. Supposing, for instance, a certain mine reckoned on 6 per cent., and it turned out to have outgoings of a million and incomings of a million-and-a-half, that would show that particular mine was making much more than 6 per cent. and had yielded one-and-a-half times as much as had been put into it.

On the other hand, another mine might have outgoings of £700,000 and incomings of £200,000. He could put the mines in order in that way, and just write a note on that which would give an answer to the President's question, that was to say, the extent of variation. It was quite an interesting point, and he was glad it had been raised.

THE LOWEST TEMPERATURE YET ATTAINED.

By H. H. PAINE.

(Printed in Journal, January, 1923.)

Mr. H. R. Adam thought members ought to express their indebtedness to Professor Paine for his very interesting summary on recent work in low temperature research. He wished to propose to him a very hearty vote of thanks. They in that Society were naturally more concerned as a rule with subjects of more utilitarian aspect than that with which the paper was concerned. However, he thought it did them good to hear of the advances being made in pure science.

There was just one point he would like to make in connection with the paper. After reading a paper like Prof. Paine's one realised that research work on the more fundamental theories of science must mean a very elaborate and costly equipment. Well, they naturally looked to Universities for research work of that description. He was sure, as a Society, they wished to see the Witwatersrand University doing what it

could in that direction also, but people did not always realise that the difficulties were almost insurmountable unless modern equipment were available. He was sure the Society would do its best to press that point with the community here. Prof. Paine's paper was so interesting to him personally that he hoped at a later date the author would give them something further on the same subject; it was clear that investigations on the properties of substances at such very low temperatures, almost at the theoretical limit, was of the utmost importance to physical science. He had much pleasure in moving a very hearty vote of thanks to Professor Paine.

ALCOHOL FUEL MIXTURES.

By H. R. ADAM.

(Printed in Journal, December, 1922.)

The President proposed a hearty vote of thanks to Mr. Adam for his paper published during the recess.

RECOVERY OF GOLD BY BLANKET CONCENTRATION IN SUBSTITUTION OF PLATE AMALGAMATION.

By F. WARTENWEILER.

Plate amalgamation has been the practice as a recovery method on Witwatersrand ores since the inception of milling. In other gold mining fields it is equally well known. Here it has been responsible for the greater portion of the total gold recovery until very recently, as is shown by the following tabulation which illustrates its relative position for the year 1921 on the Central Mining/Rand Mines Group of mines:—

RECOVERY OF GOLD BY PLATE AMALGAMATION. TABLE I.

	Per cent. of Original Value of Ore	Per cent. of Total Recovery
City Deep	68.3	70.6
Consolidated Main Reef	69.1	73.4
Crown Mines	70.8	73.1
Durban Roodepoort Deep	63.2	66.0
East Rand Proprietary Mines	54.6	57.3
Ferreira Deep	71.0	74.0
Geldenhuis Deep	61.0	64.1
Knight Central	54.1	58.1
Modderfontein B.	55.3	57.1
Modderfontein East	43.7	47.6

	Per cent. of Original Value of Ore	Per cent. of Total Recovery
New Modderfontein	64.5	66.5
Nourse Mines	61.5	64.3
Robinson	48.7	52.3
Rose Deep	58.4	62.3
Village Deep	60.3	62.8
Wolhuter	59.5	63.6

As is generally known, the variations in the above percentages of recovery are due to the differences in the degree of grinding, perfection of equipment, and the characteristics of the ore itself.

Plate amalgamation has gradually become encumbered with a number of disabilities, not the least objectionable of which are the operating cost and the numerous cases of mercurial poisoning of men directly engaged in the work. With the increasing practice of fine grinding, the perfection of cyanide extraction and of precipitation, the importance of a high recovery by amalgamation has receded according to the degree of efficiency

of the section of the reduction plant devoted to recovery of gold by cyaniding. The total elimination of amalgamation on two plants on the Witwatersrand during the past year is now history. Where, for various reasons into which I need not enter in this paper, amalgamation continues to be practised, the need for simplification has been felt for some time. No set of rules of hygiene laid down as a prevention of mercurial poisoning have been entirely effective. Developments have, therefore, been along drastic lines. The metallurgist has not had far to seek. He has found a simple method in the application of corduroy cloth to the use of gold recovery. For many years blankets of corduroy cloth have been used auxiliary to plate amalgamation, and in a few instances in substitution of the plates in the stamp mill. I trust the members of the Society will assist with the compilation of a correct historical record of the local use of blankets. According to my own recollection, corduroy blankets were used by the late Mr. Torrente at the foot of the tube mill plates at the Roodepoort United and at the George Goch plants, and in place of tube mill plates at the Meyer and Charlton as long ago as 10 years. In the *Journal* of this Society, Vol. XVIII., April, 1918, Mr. Maxwell records the adoption of corduroy at Van Ryn Estate in substitution of stamp mill plates.

The use of ordinary blanketing and special classes of fabric date back to antiquity. References to their use abound in Hoover's translation of the classic on metallurgy "*De Re Metallica*," Book VIII., and the use of a cloth in many respects similar to corduroy is mentioned in the following quotation:—"Some people use a cloth of tightly woven horsehair which has a rough knotty surface. Since these knots stand out and the cloth is rough, even the very small particles of gold adhere to it. These cloths are washed in tubs of water."

Although the use of corduroy blanketing may be said to be hoary with age, until comparatively recently, owing to the circumstances mentioned previously, it has not been used in complete replacement of plate amalgamation on these fields. There has been but scant publication of the technique and the metallurgical aspect of this practice.

With the complete substitution of plate amalgamation by corduroy blankets at a number of plants during the past year, the subject has assumed new importance, and this fact has impelled the writer, who has been associated with the development, to

place on record a description and results of recent practice and of investigations conducted in connection therewith.

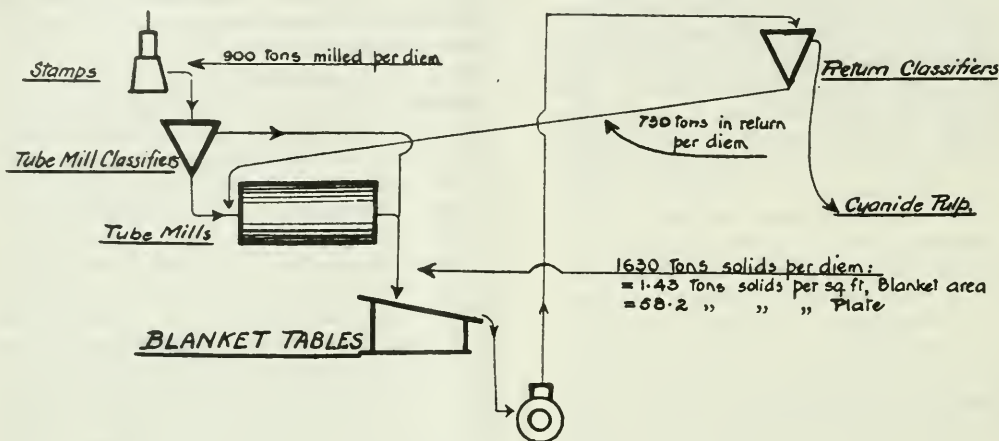
The practice at the plant treating ore from the Modder East mine, the Apex Plant, is described, it being among the first to discontinue plate amalgamation entirely. The position of blanket concentration in the ore treatment is shown diagrammatically in the accompanying flow-plan.

The amalgamating plates, freed of the accumulated amalgam from the former process, are converted to blanket tables. They are covered with the strips of corduroy cloth, placed so that they overlap each other a few inches. No holding device is used, the wet blanket clinging to the plate surface. Care is taken however to "iron out" with a spoke brush the air bubbles formed underneath when laying the blankets. It will be realised that even division and distribution of the feed pulp is important as in plate amalgamation. The area in use at Modder East is large, and was arranged to be on the excessive side for experimental purposes. Some of the other installations are using a considerably smaller area per unit of flow tonnage. As the pulp flows over the corduroy a rippling movement is observed, caused by the 3/16in. projecting ridges of the cloth. These act as riffles and the space between gradually fills with a concentrate containing the free gold and consisting very largely of pyrite. At the lower edge of the riffle projection free gold is sometimes visible. When the blanket is in good operating condition no free gold is visible on panning the tailing.

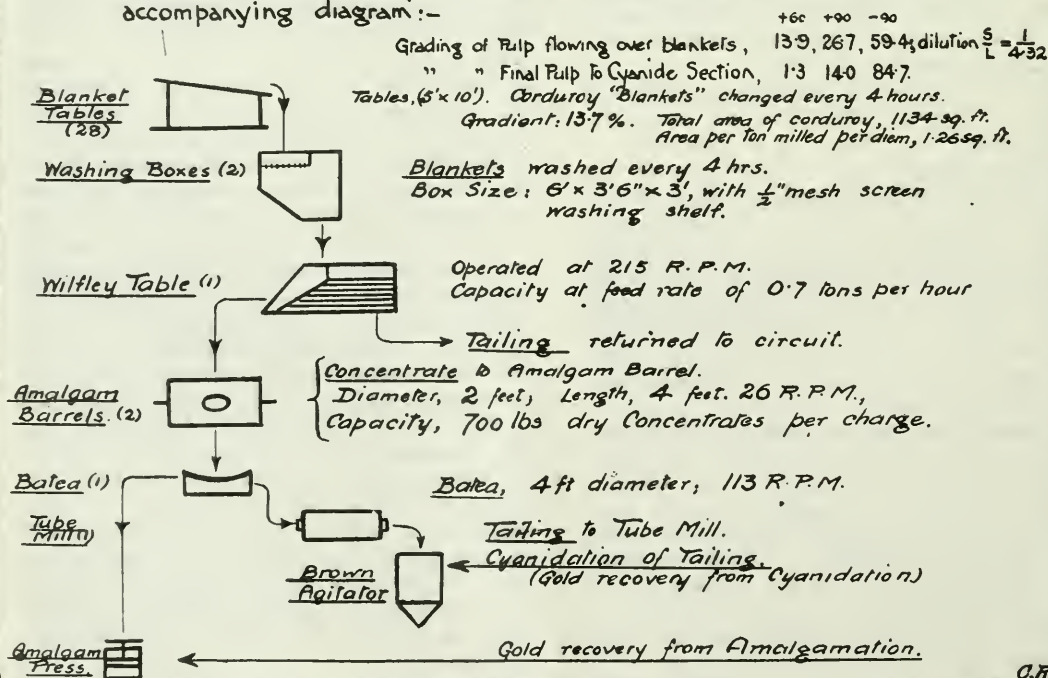
The blankets are taken off to be washed at intervals of 4 hours. Prior to carrying out this practice, the pulp is turned off and water admitted to the distributing box to wash the surface sand off the blankets. They are then folded up so as to retain the concentrate and taken to the washing box while a clean set is put down for immediate use. The washing is very simple and is effected by immersing the blanket with the riffle side down, in the water and shaking it. The concentrate readily falls and washes off, passing below the screen and settling in the box. A machine for washing is in course of construction. By having two washing boxes each day's concentrate can be kept separate.

The corduroy concentrate is fed to a Wildfey table, set rather flat and operating with a short stroke. Several definite divisions of material on the deck of the table are

— FLOW PLAN — INDICATING POSITION OF BLANKETING.



The movement and treatment of Blanket Concentrate is illustrated by the accompanying diagram:—



the common result of this operation, namely, a band of clean gold, a band of pyrite, containing also metallic iron operated from the milling machinery and the silicious tailing. At the boundaries there is a certain intermingling. An inconsiderable amount of free gold and pyrite are always found in the tailing, due to the non-perfection of concentrating tables and to floating gold and mineral carried by the stream to the tailing discharge. The tailing flows back to the milling circuit and the concentrate is collected in a box having a filter bottom.

The Wilfley concentrate is ground and amalgamated in standard type amalgam

TABLE II.

	Weight tons.	Value per ton.	Gold Content, oz.	Per Cent. of mill head, Gold Content.	Per Cent. of tons Milled.	Notes.	Grading.			
1. Mill Head (original)	27,000	10.86 dwt.	14661	100	—	{ Based on concentration recovery plus assay of cyanide pulp.	+60	+90	+200	—200
2. Final Pulp to cyanide	27,000	6.84 "	9234	63	—					—90
3. Nett concentration	—	4.02 "	5427	37	—	{ Assay of pulp after concentration.	1.3	14.0	—	84.7
4. Corduroy concentrate	39.7	138.3 oz.	5493	37.4	.147	—	6.7	22.2	31.2	39.9
5. Wilfley tailing	18.3	3.6 "	66	0.4	—	Returned to circuit.	6.5	21.9	37.2	34.4
6. Amalgam	15,865 oz.	32%	5077	34.6	—	93.7% of concentrate recovery.	—	—	—	—
7. Barrel tailing	21.4	16.3 oz.	350	2.4	—	{ Cyanide extraction on this is 97.2%.	1.2	2.0	21.4	75.4
8. Cyanide residue	21.4	9.1 dwt.	9.7	0.1	—	{ Equivalent to 0.007 dwt. per ton milled.	97.5%	—200 mesh.	—	—
Recovery by cyaniding of barrel tailing	340.3	2.3	—	6.3% of concentrate recovery.
Total net recovery by concentration	5417.3	36.9	—

It will be noted that the recovery by the blanketing concentration method is 36.9 per cent., of which amalgamation accounts for 34.6 and cyaniding for 2.3 per cent. This compares with 41.8 per cent. obtained previously by plate amalgamation. Mercury consumption has been reduced by 66 per cent. and is now .05 oz. per ton milled. Cyanide residues have not been affected and total extraction remains the same. A study of the grading analyses discloses the comparatively fine condition of the concentrate caught on the blankets. On this plant, milling 900 tons daily, the final concentrate which is amalgamated is .71 tons, or .079 per cent. of the tonnage milled. The Wilfley concentrate consists of free gold, mineral sulphide, metallic iron and silicates. As previously referred to, these divisions are readily observed as bands on the concentrating table in the order of their specific gravity. The mineral sulphide shows in two distinct divisions, one containing pyrite (FeS₂) and pyrrhotite, the other consisting

barrels, after which the amalgam is separated in a batea in the usual manner. A period of 14 to 16 hours is allowed for grinding and 2 hours for amalgamation. The tailing from the barrel amalgamation is reground in a small tube mill and cyanided. These last two steps, in the case of Modder East concentrate, are carried out at the Witwatersrand Co-operative Smelter treatment plant.

In the following tabulation, Table II., is shown the particulars of gold recovery in the different stages. The data are a typical illustration of the incidence of weights, values and recoveries over one month:—

of the heavier minerals and metallic iron and having a bluish appearance in mass. An analysis of the bluish mineral carried out two years ago is as follows:—

Arsenic	15.6 per cent.
Antimony	3.0 per cent.
Cobalt	13.1 per cent.
Nickel	5.0 per cent.
Sulphur	26.0 per cent.
Iron	22.3 per cent.
Phosphoric Acid	13.1 per cent.

98.1 per cent.

An analysis of the Modder East screen sample was published in the May, 1921, number of the *Journal*. Its re-publication here with regard to some of the above constituents is of interest in indicating the degree of concentration on the corduroy:—

Arsenic sulphide	0.020 per cent.
Nickel sulphide	0.036 per cent.

The mineral sulphide content of the ore in terms of FeS₂, and calculated from the

sulphur content is 1.94 per cent. In 1,000 tons of ore the sulphide content is therefore 19.4 tons. Of this approximately 0.67 tons is collected by the corduroy.

Recent laboratory investigation has increased our knowledge of the gold distribution in the Wilfley table concentrate. A sample of this concentrate, in this instance assaying 450 ozs. gold per ton (the difficulty

of securing an average sample will be understood) was divided first by amalgamation into free gold and concentrate. It was then further classified by vanning and elutriation into heavy sulphide, pyrite, quartz and silicates. The gold distribution was found to be as shown in Table III. It may be taken as a close approximation on this difficult material or to be within determinative limits:—

TABLE III.

	Per Cent. by weight.	Assay Value Oz. per ton.	Oz. Gold per ton Concentrate.	Per cent. Total Gold.
Wilfley Concentrate	100	—	397.8	100
1. Free Gold (with alloyed silver)	1.2	—	388.8	97.73
2. Heavy sulphide (consisting of nickel, cobalt, and arsenic sulphides, also oxides of chromium and vanadium)	2.6	41.5	1.1	0.27
3. Pyrite (consisting of pyrite and pyrrhotite)	83.1	8.3	6.9	1.73
4. Quartz and silicates	13.1	7.6	1.0	0.26
	100.0		397.8	

The extremely large percentage of value in the form of free gold is remarkable, and the necessity of further research on this point is indicated. It would appear that practical barrell amalgamation falls short of amalgamating all this gold.

The metallic iron being extremely fine and flaky, oxidised rapidly and is included in the classified products. Its value therefore was not determined separately. Considerable difficulty was experienced, even after amalgamation, to liberate the classified products entirely of free gold, and this probably accounts for the unexpected high gold content of the quartz and silicates.

The gold taken off clean from the Wilfley table is found to be extremely fine. A

grading through sieves shows it to be:—

— 100 +200	— 200 +260	— 260
0.53%	9.05%	90.42%

Examined microscopically, a few grains show flattening as by a blow, a few show striations such as may be caused by rubbing against a rough hard surface, but the greater portion appears in a crystalline and nuggety form. The grains range in size between 0.2 mm. and 0.02 millimetre in cross-section.

From the beginning experiments have been continuous in order to obtain more accurate knowledge and with the view of arriving at the best practice to suit the circumstances. The distribution of free gold on the various blankets from the top to the bottom of the table, after a 4-hour run, has been found by panning and weighing of the free gold, on different days, to be as follows:—

Collects these percentages of the total.

	%	Cumulative %	%	Cumulative %	%	Cumulative %
No. 1 Blanket	75.97	75.97	79.79	79.79	79.58	79.58
No. 2 Blanket	17.83	93.80	11.70	91.49	10.47	90.05
No. 3 Blanket	3.10	96.90	3.72	95.21	4.19	94.24
No. 4 Blanket	2.32	99.22	3.19	98.40	3.66	97.90
No. 5 Blanket	0.78	100.00	1.60	100.00	2.10	100.00

It would appear from the above that the lowest blanket (No. 5) and even the two above might be discarded, according to the amount of gold one was prepared to pass to the cyaniding section. The presence of a high value mineral sulphide on the fourth and fifth blankets, however, calls for consideration. This mineral has been separated from the siliceous portion and from the free gold. It assays as follows:—

	Average Assay. Oz. per ton clean mineral.
No. 4 Blanket	334
No. 5 Blanket	280

It was found that these last two blankets ceased to function as concentrators after 5 hours' operation.

(The lock-up or absorption of gold by the corduroy was determined and found to be very small. The gold content of a washed blanket after several months' use was ascertained to be only 0.20 oz., or 28 oz. over 28 tables. This small absorption has been confirmed by the realisation of the gold from an accumulation of discarded blankets.) No elaborate experiments with other concentrating surfaces were carried out on this plant as this subject had been investigated during a previous period. A few tests with jute resulted rather unfavourably.

Some years ago, when a scarcity of mercury was imminent, experiments were carried out on stamp mill plates with different types of concentrating cloth. It was found then that corduroy and mill blanket gave about the same gold recovery, but the bulk of concentrate was greatly less with corduroy. The concentrate from a canvas surface was the least bulky, but the gold recovery was only 72 per cent. of that with corduroy. Ordinary riffles were 57 per cent. as effective in recovery as corduroy.

The question of treatment of the Wilfley concentrate from the blanketing was at that time also studied at the New Modderfontein plant.

In experimenting with the concentrate, including the free gold, results were as follows under various treatments:—

1. Ground -200 mesh and cyanided. Extraction 89.5%.
2. Ground -200 mesh, amalgamated, 79.3%; then cyanided. Total extraction 99.7%.
3. No grinding, only percolation with cyanide. Total extraction 28.7%.
4. Roasted, ground -200 mesh, amalgamated, 83.4%, then cyanided. Total extraction 99.7%.

After separating the free gold with the Wilfley, the following experimental results were obtained:—

5. Ground -200 mesh and cyanided. Extraction 88.1%.
6. Roasted, ground -200 mesh and cyanided. Extraction 70.3%.
7. Roasted, ground -200 mesh, amalgamated, 92.9; then cyanided. Total extraction 99.9%.

The gold loss during roasting was high, and no particular improvement in extraction was secured by it, also no advantage appeared to be gained by first separating the free gold. Amalgamation appeared to be a necessary step in the treatment. Smelting did not offer a suitable process without a preliminary roasting or sintering.

The advantages of the method described in this paper over plate amalgamation have generally fulfilled expectations. The amalgam room staff only is now handling amalgam, thus definitely and considerably reducing the risk of mercury poisoning. The personal element of skill and care in dressing amalgamating plates has been eliminated. One cannot help speculating whether this will become one of the lost arts. Amalgam is no longer found in the tube mill circuit, and any uncertainties arising from this source trouble the reduction officer no longer. As the recovery of the gold from the concentrate is regular daily practice, and not dependent on conditioning amalgam plates, the daily recovery reflects closely, variations in screen or mill head value, and therefore serves as a check. On the economic side, simplification of the work and the saving of interest on gold formerly absorbed by amalgamating plates and by amalgam in the circuit are the principal factors.

One may assume that the practice of concentration in substitution of plate amalgamation will experience further development in respect of technique, in research for a more permanent riffle material and in mechanical arrangements. Further experiments will undoubtedly be conducted with the object of cyaniding the corduroy concentrates in order that amalgamation may be discontinued entirely.

Acknowledgment is due to the manager of Modder East, Mr. E. Pam, for permission to publish various data contained in this paper; to Mr. Fiander, reduction officer of the Modder East Co., for his keen interest and cordial co-operation in developing the practice; also to the Rand Mines Laboratory for the chemical analyses.

Lt.-Col. E. Pam said the mine manager's attitude on a metallurgical question like that was necessarily brutal. The technical points were, of course, of certain interest, but the main point which was asked was, was the extraction going to be similar or better, and the cost going to be any more?

When Mr. Wartenweiler suggested the scheme at first a start was made with a few plates, but within the last few months all amalgamation on plates has been entirely abandoned. The most important fact from the manager's point of view was that the final residues had not gone up. As far as economy was concerned there was no proof that much money was being saved. There were, however, indirect economies which could not be calculated, but which were, nevertheless, important. One could quote, of course, the question of health, in which the Society had always taken a great interest. The fact that a smaller number of men handled mercury was obviously an advantage.

The author did not mention the question of theft, but it must be agreed that if, by this process, they put temptation in the way of fewer people they were obviously doing the right thing, quite apart from a possible saving in gold.

Another point of importance to the mine was the question of no longer having any amalgam to fall back on at the end of the month; that, from the manager's point of view, was not altogether an advantage. (Laughter.) On the other hand, economically it was obviously sound, and no doubt managers would carry on somehow.

He proposed in their name a hearty vote of thanks to Mr. Wartenweiler for the paper over which he had clearly taken a lot of trouble, and it was certain that those who read the paper carefully afterwards would see that it was worthy of the best traditions of the Chemical, Metallurgical, and Mining Society of South Africa.

Mr. H. A. White said he would like to second the vote of thanks for the very excellent paper, so full of valuable details, and one which, of course, would be of peculiar interest at the present time when so many mines were considering the question of whether it was absolutely necessary to maintain that doubtful practice of amalgamation which had been responsible for considerable sums of money being paid in compensation for alleged mercurial poisoning

every year, and which was the base of most of the gold thefts reported. He could not help thinking, however, that the introduction of corduroy concentration was only one step, and the next and logical step, after getting the full advantages of introducing it, would be to get the fuller advantages of throwing it out. (Laughter.) He could see no real reason, and none was visible in the figures before them—and, listening as carefully as he could, he could not notice any in the paper read before them—why the method of recovering a portion of the gold in the mill should not be abandoned entirely, and there seemed to be little reason left under present conditions why they should not rely upon the cyanide works for getting the whole of their gold. He believed it would be found at those mines which had the pluck to take that necessary whole-way step, that there would be not only no loss by higher residues, but that in all cases the residues would be slightly reduced. It was certainly the case at Springs; he knew it to be the case in one or two other mines in that district that the residues were distinctly decreased, although not by a great amount. That, he supposed, might be due to the fact that all the pulp which left the mill, and even gold which escaped in the barrel residues, carried mercury quite sufficient to cover the whole of the exposed gold surfaces with mercury. Exactly what effect that may have upon the solubility of the gold he did not know; but he proposed to make some experiments on the point. That was the only reason he could see at present for the undoubted fact that wherever amalgamation was totally dispensed with there was some improvement in the total residue.

There was one little point in connection with the paper in which he noticed Mr. Wartenweiler introduced a rather elaborate explanation—that was, as to the value in the second column of the figures on the blackboard where they saw pyrite, consisting of pyrite and pyrrhotite value 8.3 oz., while quartz and silicates showed 7.6 oz. They were too nearly alike in value to suit the preconceived notion which still prevailed on the Rand, with no experimental basis, namely, that there was a close association between the pyrite and the gold. Those figures as they stood did not bear out this assumption at all, and he thought the necessity for that *ad hoc* explanation was simply the fact that the gentleman who was responsible for it was perfectly certain in

his mind that there was close association between pyrite and gold in the ore and could not account for the fact that it did not show up in his experiments.

He had very much pleasure in seconding the vote of thanks to Mr. Wartenweiler for his unusually valuable paper.

NOTES ON CUPELLATION LOSSES.

By H. R. S. WILKES.

Mr. H. R. S. Wilkes, in introducing his paper, said he had a certain amount of trepidation in bringing these Notes on Cupellation Losses before members, but his idea was to try to settle what appeared to be a debateable point among assayers. The matter had been settled in his own mind for some considerable time. He would like to know for certain whether other assayers thought alike with him.

To attempt to find anything new to say about cupellation would appear presumptuous. The writer, therefore, in presenting this note will not be surprised to be told that the whole matter was settled and pigeon-holed long ago. He offers as his excuse that he has never found the particular pigeon-hole in which it has been stored.

Losses in cupellation are spoken of and accounted for by airy mention of the processes of absorption and volatilisation. While it is easy enough to demonstrate that absorption does take place, it is quite another matter to prove loss by volatilisation.

It probably will not be disputed that cupellation losses only take place during active oxidation, and that when all but traces of the base metals have been got rid of, that is, when cupellation has ceased, no further loss takes place.

The late J. J. Beringer in his classical work on assaying states on page 101: "The loss of silver by volatilisation is altogether inconsiderable unless the temperature at which the operation is performed is much too high."

It is a distinction, perhaps with very little difference, to state that loss of the precious metals can be caused by volatilisation of the lead; it certainly does not take place by volatilisation of the precious metals themselves.

When rapid volatilisation or fuming of the lead is going on the fumes can, and

sometimes do, carry with them extremely minute particles of the lead alloy bath, from which particles as they pass away with the fume the lead portion rapidly oxidises out, the infinitely minute residual particles of silver and gold alloy being carried along in the fume stream to be deposited on flue surfaces. The writer submits in support of this that he has assayed portions of used-up muffles taken from close around the back vent hole and found traces of silver and gold.

The writer has exposed cupelled beads to as high a temperature as is obtainable in a muffle for as long as five hours, and has detected no further loss after cupellation has ceased. "Neither gold nor silver will volatilise at muffle temperatures." (T. Kirke Rose, this journal, 1905, p. 49.)

The loss by absorption is caused in much the same way, and is purely mechanical; it is the cause of a much greater loss than that by volatilisation (so called), because the vehicle which carries off the particles of lead alloy is the more viscous molten litharge. This, constantly forming and rushing off the convex surface of the lead bath to the edge, carried with it lead alloy particles, which if minute enough enter the porous surface of the cupel where complete oxidation of the lead takes place, leaving the residual precious metals imbedded.

In this connection a curious phenomenon is noted; the alloy of silver and gold which is recovered from a smelt of the cupel is always higher in percentage of silver than that forming the assay bead. Mr. J. J. Beringer remarks with regard to this, "There can thus be little doubt that the greater part of the cupellation loss is a chemical one."

The writer ventures to disagree with this conclusion, unless the professor considered "liqutation" to be a chemical reaction, instead of merely the action of gravitation. Liqutation in molten alloys is the partial separation of the metals of higher specific gravity from those of lower.

Examining this phenomenon of liqutation in its bearing on cupellation, we have a molten bath of alloyed metals in a state of almost perfect quiescence, a condition favourable to liqutation.

In this bath there is a constant tendency of the metals of a lower sp. g. than lead to rise to the surface and those of greater sp. g. to sink to the bottom. This separation is of course imperfect, but it has marked effect on cupellation losses; the result of this partial separation is to set up an enrichment of the surface of the bath in silver and the

reverse with regard to gold, since the loss of precious metal takes place always from the surface of the bath it follows that the absorption of metal into the pores of the cupel will result in an alloy high in silver percentage rather than gold, thus the cupellation loss of silver will exceed the percentage of the silver in the original alloy of gold and silver, and the finished bead will show a much lower percentage of silver than gold, than that of the alloy obtained by smelting the cupel.

The foregoing is to a certain extent confirmed by observations and assays made of litharge sampled as it came off the "tests" used in the cupellation section of the pan furnace method of smelting zinc-gold slime, which is the system most in use on the Rand.

The figures given below are from notes made as long ago as 1909. The lead bullion from the pan furnace gave an original assay value of 5.70% F.G. and .63% silver, a ratio of gold to silver in the bead resulting from the assay of 88.95 gold to 11.05 silver.

Time of sampling.	Value of Litharge.		Ratio of	
	F.G.	F.S.	G. to S. in bead.	
	dwts. per ton (2000 lbs.)		Gold.	Silver.
1st hour ...	14.00	29.00	32.56	67.44
2nd hour...	17.00	30.00	36.16	63.17
3rd hour...	48.00	63.00	43.25	56.75
4th hour...	45.00	60.50	42.66	59.34
5th hour...	127.00	131.50	49.13	50.87
6th hour...	185.00	75.00	70.00	30.00
7th hour...	650.00	125.00	84.00	16.00

The "test" broke down shortly after taking the seventh sample. These figures show an almost constant decrease in the percentage of silver in the assays of the litharge as the lead bath increased in percentage of precious metals, but all these results give a higher percentage of silver than the finished bullion contained, which was 8.30%.

The following are results obtained in present-day practice of assays of litharge produced by cupellation of pan furnace lead bullion; the figures give the percentages of gold and silver in the beads resulting from assays of the litharge:—

No.	F.G. %	F.S. %
1	55.3	44.7
2	54.9	45.1
3	82.1	17.9
4	71.6	28.4
5	56.8	43.2
6	57.3	42.7
7	53.2	41.8

8	72.9	27.1
9	73.7	26.3
10	54.7	45.3

In all these results the silver is greatly in excess of that in the finished bullion resulting from the cupellation.

In the case of a small quantity of copper being present in the lead alloy it is well known that all but traces of copper works off in the early stages of cupellation, copper having a lower sp. gr. than those of the other metals, i.e., 8.82.

There is another theory which is advanced to account for the loss of silver in excess of gold during cupellation.

It may be briefly stated as follows: "The loss of the precious metals by volatilisation and absorption takes place as oxides, and as silver oxidises more readily than gold a greater amount of it passes off with the litharge than of gold."

In support of this theory it may be stated that Wait (Transactions, Am.I.M.E.) extracted by means of acetic acid about 19% of the total silver in litharge (total was 2.94%), and as metallic silver is insoluble in acetic acid this proves that silver was present in the litharge as oxide.

Since, however, only 19% of the silver present existed as an oxide, something more than oxidation must account for the excessive silver loss, and the writer submits that the principal factor is liquation.

In conclusion the writer will quote an instance which would appear to corroborate the liquation theory.

Towards the finish of a cupellation of a batch of pan-furnace lead bullion a small crack developed in the breast of the "test," which communicated with the bottom of the bath of alloy. The litharge which was coming off at the time was assayed and gave the following result:—

Gold and silver bullion, 1577.00 dwts. per ton.

Fine gold, 1398.00 dwts. per ton.

Fine silver, per cent. in bullion, 11.4%.

This silver percentage approximated closely to the silver percentage in the bullion resulting from an assay of the metal leaking through the crack.

Mr. A. Whitby, in proposing a vote of thanks to Mr. Wilkes for his paper, wanted, at the same time, to point out that, with regard to the assay of gold, more particularly in reduction samples, the very point raised by the author was brought forward

by him (the speaker) many years ago. He had then pointed out that there was sometimes diffusion and often liquation on the surface of the cupel, especially when the button of lead was too large, that was to say, there was a scorification effect, and, as a result, the silver went into the cupel; and it had often happened to him, and he dared say to hundreds of assayers, that there was no silver left, but merely a prill of gold. If they examined it closely with the lens, and no assayer should be without a lens, they would find microscopic prills all over the green surface, mostly of silver; they were retained in the cracks of the cupel.

Mr. J. Watson had much pleasure in seconding the vote of thanks to Mr. Wilkes for his interesting paper. Silver being worth only about 3s. per ounce and gold being worth £4 5s. or upwards, the custom on the Reef, where they had only about 10% of silver in the bullion, had been to run the gold assay as accurately as possible: but the silver had been more or less allowed to rip, he was afraid, on many Rand Mines. He had always considered that silver was more easily oxidised than gold. Of course, with a large bead as in bullion assays one saw the "peacocking" effect described by Sir Thomas Kirke Rose and others; but with the smaller beads they did not see this effect. He was afraid, with a batch of, say, twelve cupels in the muffle, it was sometimes the practice of assayers to leave the cupels till the whole batch was finished. He personally made a practice of getting out each bead very shortly after the cupellation was finished, in cases where silver was being determined.

He had been told that in the United States, when they were cupelling for silver, they ran the temperature of their muffle very low, so that they got around the edge of each cupel the feather flakes of litharge. Here, on the Rand, they ran their muffles at a much higher temperature, as a rule. He had much pleasure in seconding the vote of thanks to Mr. Wilkes.

Mr. A. Whitby said the late Mr. William Bettel pointed out to him quite a number of years ago that, in cupelling with the ordinary bone-ash cupels, when the silver was added for inquantation, it was necessary to keep the temperature down to the lowest and so get the feathering of the litharge that Mr. Watson had mentioned. It was a little intricate point in assaying. He quite agreed with Mr. Watson that, "if you are

inquanting keep your temperature as low as possible, and do it in a bone-ash cupel in preference to one of any other kind."

Mr. H. L. Krause asked if Mr. Wilkes would be good enough to explain to those of them who were more interested from the point of view of mine assays whether that loss of silver was of sufficient importance to warrant any investigation of their methods where they deducted approximately 10% for the silver in all their mine assays, or whether they were erroneous in that, or likely to be excessive in error? He did not know whether his experiments would assist them on the point.

Mr. A. Whitby asked whether he might answer the question.

The President replied that he might if Mr. Wilkes were agreeable.

Mr. Wilkes said he was quite agreeable; he would like to hear Mr. Whitby's explanation.

Mr. A. Whitby said the whole question was this. Every assayer should examine his cupel closely with the lens when he took the cupel out of the fire. He had to see that his bead stood out clearly and that there was no green ring around the bead. If he found it he could be pretty well sure there was silver and sometimes gold absorbed.

Mr. Wilkes said, not quite agreeing with Mr. Whitby, he would like to answer Mr. Kruase's question himself. The question of scorification in the cupel was one which, he thought, the active and intelligent assayer did not worry about. As a matter of fact true scorification never did take place. But if the surface of a cupel became glazed it simply increased the length of time of the assay, for by fuming and also forming lead silicate on the muffle floor the cupellation would be completed eventually. In any case, however, that result would be valueless, and the assay should be repeated.

The estimation of silver in mine samples, in his opinion, should be made by a careful bullion assay of a collection of mine sample beads, gathered by the assayer as soon as possible after his appointment to a mine, and when he has satisfied himself that under ordinary working conditions his finished mine beads contain, say, 8, 10, or 12½% of silver, as his result may determine, any

absorption which may have taken place need not concern him, he can look upon the percentage of silver found as his factor, and the amount of difference any slight error might make to his stope values or assay plan will be negligible.

Mr. A. Whitby was sorry to say he still disagreed with Mr. Wilkes, because he had not observed the special characteristic to which he (the speaker) had referred, that was, the green enlargement underneath the prill in a cupel; Mr. Wilkes attached little significance to it; and the speaker did not believe there were many assayers who gave it sufficient consideration.

Mr. Wilkes said he desired to take up the cudgels for mine assayers on the Rand. He did not believe there was a single assayer along the Reef who had not noticed that green stain. Perhaps some of them had not tried to find out what caused it. He would like Mr. Whitby to tell them what form of gold or silver imparted a green stain to a cupel.

Mr. A. Whitby replied there was no reasonable explanation of the colour, so far as he knew. Mr. Wilkes had evidently also found it; so he was glad he had confirmed him.

NOTE ON THE CONTAMINATION OF COKE SAMPLES WHEN PREPARED IN A DISC GRINDER.

By J. HENDERSON.

A trial truck of coke was received by the Witwatersrand Co-operative Smelting Works, which was carefully sampled and the sample prepared in the apparatus available, namely, a small jaw rock breaker, an Allis Chalmers coffee-mill-type sample-grinder and finished to -90 mesh in a "Braun" disc pulveriser. The ash results were much higher than was expected, particularly as an up-to-date washing plant had only recently been installed at the coke works. Knowing the hardness of coke and its capabilities of scratching even steel plates, the grinders were suspected as a possible source of contamination. A further sample was taken, broken down to $\frac{1}{2}$ inch, thoroughly mixed and divided into two portions. One-half was prepared as before and finished in the

disc grinder. The other half was "pounded" down in a large Wedgewood mortar, but not "rubbed" or "ground." The ash results were as follows:—

Disc grinder portion	13.35%
Wedgewood mortar portion	12.15%

Difference	1.20%
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A further consignment was sampled and treated as above. The ash results were:—

Disc grinder portion	12.90%
Wedgewood mortar portion	11.75%

Difference	1.15%
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It was easy to see that there was more oxide of iron in the ash from the disc grinder portion than in the other.

The two ashes from the second experiment were tested for iron. The results calculated as percentage on the coke were:—

Disc grinder portion iron	1.8%
Wedgewood mortar portion iron	1.0%

Difference	0.8%
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0.8 of iron is equivalent to 1.14 of ferric oxide, and accounts for the difference of 1.15 in the ashes.

The results clearly show the heavy cutting action of coke particles and the danger of reducing samples of coke to a fine state of division by grinding in disc grinders.

The general method used in public and work laboratories in the North of England iron district was to finish coke samples in a heavy steel crushing mortar, such as is used for pounding samples of white iron and ferro manganese. Throughout the preparation of the sample the coke was broken down by percussion only, and in no case "rubbed" or "ground."

Since the above note was sent in I came across an extract in one of the volumes of the Society dealing with contamination from sample grinders. The extract will be found in Vol. 13, page 267, and may be of pertinent interest to the members.

A bed of quartzite was under examination as to its suitability for the manufacture of silica bricks. As oxide of iron is an undesirable constituent for high-class silica bricks, the correct determination of the iron contents of the quartzite was of the utmost importance. The samples were ground to -120 mesh in disc grinders. On being tested it was found that serious contamination had occurred by the cutting action of the quartzite on the discs of the grinder. The

increase in the iron due to this contamination amounted in some cases to nearly 3%. If the contamination had not been detected the bed of quartzite would have been unjustly condemned for too high iron contents.

My thanks are due to the Witwatersrand Co-op. Smelting Works, Ltd., for permission to publish this note.

Mr. J. A. Woodburn said he would like members to pass a hearty vote of thanks to Mr. Henderson for his very important note in connection with the investigation on coke. He thought that notes on practical work such as the one Mr. Henderson had brought forward were most valuable, because they encouraged others to look into various points of practical experience which were often overlooked.

SYMPOSIUM: MINERS' PHTHISIS.

Mr. H. J. Ireland: The contribution by Mr. J. D. Marquard in the September number of the *Journal* raises questions to which perhaps I can suggest some explanation.

I was interested to read Mr. Marquard's statement "that in the deeper mines, such as the Jupiter and Simmer Deep, men developed silicosis sooner than in the Geldenhuis Deep or the Rose Deep." It is what one would expect, though not exactly for the reasons he gives, the reason being, in my opinion, that at the greater depths the cooling power of the air is much lower, and inadequate for maintaining normal conditions of the body.

There has been in recent years a good deal of evidence forthcoming from various sources to the effect that hot, humid conditions tend to increase the damage caused by the presence of fine dust. So far as I am aware, there has been little or no attempt to analyse the physiological causes of this, and I offer here some explanations of the fact.

Let us examine the various effects produced by hot humid air on a mine worker.

We have (1) the direct effect of very low cooling power, resulting from high temperature, high humidity, and lack of motion, causing severe strain on the heat regulating mechanism. To quote Dr. Leonard Hill, "blood is pooled in the cutaneous capillary area, which should pass in ample stream through the brain and muscles. These may lack oxygen and nutrition needed for

strenuous exertion and exhaustion results as the heart becomes fatigued under the double strain of keeping up the work and cooling the body."

This pooling of the blood in the dilated cutaneous blood vessels is equivalent to a withdrawal of a large quantity of blood from circulation, consequently there is a decreased flow through the lungs and, as a result, a lower respiratory efficiency, by which is meant a smaller percentage of oxygen is transferred to the blood and a greater ventilation of the lungs required; the more rapid action of the heart and breathing muscles still further increases the lung ventilation to supply the increased need of oxygen. This increase of the volume of air inhaled will undoubtedly produce an increase in the dust inhaled and deposited.

Then we have (2) the direct effects of humidity. If the air inhaled has a large moisture content, the evaporation from the respiratory tract is small. As evaporation from this tract is replaced by a flow of fluids from the lymph glands and other tissues which tend to prevent the lodgment and penetration of bacteria and other foreign bodies, the disadvantage of a small evaporation is obvious.

A large moisture content will, of course, produce a lowering of the wet kata cooling power with the effect described in (1), but apart from this there is a good deal of evidence from the observations, not only of physiologists but practical men like D. Harrington of the U.S.A. Bureau of Mines, and Dr. H. Briggs of Edinburgh, to show that humidity has a direct effect in impeding the transference of oxygen to the lungs, consequently lowers the respiratory efficiency and increases the amount of lung ventilation necessary.

Then we have (3) the indirect effects of low cooling power. It has been shown repeatedly that low cooling power induces fatigue, inhibits the elimination of toxins from the system and lowers the vital energy. In all probability this decreases the natural defences of the lungs against the inroads of dust.

For the above reasons it is evident that "cooling power" as such is a considerable advantage, and in arranging methods of ventilation to dilute or eliminate dust at the workings it is important to provide for the maximum "cooling power" consistent with other needs. As Dr. Orenstein and I have pointed out before, where the normal

movement of air due to the ventilation currents is not of sufficient velocity to give the desired cooling power, recourse should be made to means of producing local movement, not as a substitute for change of air, but as an accessory to it.

I once had the hope that with a large fresh air supply to dilute the dust-laden air it would be possible to keep the air comparatively dry and gain the advantages to health which dryness gives, but a little application of elementary physics to the problem convinced me that in deep mines at least this was a vain hope. As the rock temperature increases about 4° Fah. per 1,000 feet increase in depth and the temperature of dry air as it is forced to lower levels increases 5.3° Fah. for the same increase in depth due to compression, it is clearly impossible to cool a deep mine with dry air, and we have to fall back on evaporation of moisture to cool the air sufficiently to absorb heat from the rock.

What can be done, however, is to trap dust, and lay dust, and where these fail, dilute dust by ventilation, and increase the cooling power by movement of air as much as possible. Special attention should be given to adequate distribution of air to the lower levels which require a much greater share of fresh air than the upper levels, not only because of the greater quantity of heat emitted from the rock, but also because each pound of air circulated will absorb less heat at the lower level due to its temperature being raised by natural compression. This is an aspect of the problem which, it seems to me, has not received the consideration it deserves and which is of supreme importance in the ventilation of deep mines.

To come back to Mr. Marquard's problem, I think then that the reason why silicosis is more prevalent in the deeper mines is that in the first place it is somewhat more difficult to get good ventilation at the deeper levels, and, in the second place, even with the same ventilation, one cannot get anything like so great cooling power.

I don't see how there can be much, if any, disadvantage in compressed air as such, except that, as a means of ventilation, it is expensive, and therefore limited. With regard to finely-divided oil in the compressed air I take it that most of this oil is mineral oil of the paraffin series, and would point out that the various antiseptic oils such as oils of cinamon and eucalyptus, are administered in a fine spray diluted with paraffin oil for the treatment of bronchitis, catarrh,

and other respiratory troubles. Moreover, there has been developed in recent years a method of treatment of wounds and inflammatory disorders called dielectric therapy, which employs a standard paraffin oil as the curative agent, the oil being supposed to penetrate the tissues and act as a protection to the cells against bacteria and their toxins. I don't think, therefore, we need have much fear of harmful effects from the minute quantities of lubricating oil which find their way into the compressed air.

GOLD PRECIPITATION BY ZINC DUST AND IN CONJUNCTION WITH THE DE-AERATION OF SOLUTION AT MODDERFONTEIN "B."

By S. NEWTON and L. L. FEWSTER.

(Printed in Journal, June, 1922.)

REPLY TO DISCUSSION.

In replying to Mr. White's contribution to the discussion on the above paper, we should like to correct two errors which crept into the text as printed.

(a) Run No. 22 should read "19 tons per filter leaf per 24 hours" as two presses were in operation for this run only. The corresponding figures in all other runs are correct; it should be remembered that "per filter leaf" is taken as indicating "per double-sided filter unit or per hollow frame with its two adjacent filtering surfaces."

(b) The "amount of lead nitrate added," as appearing in the tabulation, is correct, and in the text it should read "24 lbs. per 24 hours."

This would mean according to Mr. White's calculation that 900 lbs. of metallic lead added to the solution to be precipitated is represented by 1,637 lbs. of metallic lead in the precipitate taken from the press, the difference of 737 lbs. being derived from the lead nitrate used on the sands and slimes plants.

(c) We are in agreement with Mr. White that the amount of zinc remaining in the press might profitably be reduced, but so far we have not had the opportunity to determine the minimum zinc feed.

We had hoped by this time to have made considerable progress in this direction, but on account of having received a large consignment of damaged zinc dust we have been

compelled to return to the use of excessive amounts of precipitant to ensure satisfactory results as regards filtrate values. A margin of safety in the press is not only desirable but essential, as evidenced by the tabulation on page 253. Tests 1 and 2. As regards the present practice of direct smelting of the Merrill press precipitate, we admit that the product is not ideal, but it is certainly suitable for such treatment by or in the Tavenor furnace.

A great reduction in the excessive amount of zinc remaining in the press would possibly suggest acid treatment and pot smelting.

(d) As regards zinc dust feed at 4 lbs. per 100 tons of solution as mentioned by Mr. White, this we may say has been anticipated, but for the reason mentioned above its application has not been permissible. This remark applies to the consumption of lead salts.

As regards the necessity for maintaining cyanide strengths above .017% KCN. we wish to state that this was a purely local detail governed as far as we could see by the other purely local conditions, such as alkalinity and dissolved lead contents; and amount of zinc feed.

The determination of what may be possible by means of variation of these factors has not been lost sight of, but has been prevented by the prevailing conditions.

We wish to thank Mr. White for his very interesting contribution, especially his closing remarks on the solubility of lead salts which certainly might be investigated by some of our chemical members to our mutual advantage.

The authors also wish to thank Mr. Wartenweiler for his contribution from the "cost" standpoint. Mr. B. J. Hastings has also added to the fund of knowledge with his contribution to the discussion.

INVESTIGATIONS ON THE COPPER NICKEL ORES OF THE RUSTENBURG DISTRICT.

By J. A. ORTLEPP.

(Printed in *Journal*, August, 1922.)

DISCUSSION.

Mr. John Watson: The fact of this paper being published in the August number during our winter recess, and not read at a meeting, may partly account for the sequence that, up to date, there has been

no discussion. The paper shows much careful and painstaking work, on the author's part, in the preparation of sections for microscopic examination, six microphotographs, with descriptions, being printed.

Mr. Ortlepp also describes the metallurgical investigation, concentration, and flotation tests. By matte smelting he was able to enrich the nickel content to 25.30% in the matte. The records of such experiments are most valuable for reference, as some of us may find ourselves up against a similar problem sooner or later.

I have much pleasure in moving a hearty vote of thanks to Mr. J. A. Ortlepp for his interesting paper.

SOME NOTES ON THE EVOLUTION OF THE KONIMETER.

By R. A. H. FLUGGE-DE SMIDT.

(Printed in *Journal*, November, 1922.)

DISCUSSION.

Mr. S. de Smidt: I was somewhat astonished at Mr. Flugge-de Smidt's statement about the results of Mr. Boyd's experiments and the doubt expressed on the work of Drs. Moir and Mavrogordato. The results of their experiments have not been published as far as I know, but they are sufficiently well known. The results of Mr. Boyd's experiments are also against the experience of the officers of the Mines Department.

I would like to suggest to Mr. Flugge-de Smidt to try and persuade Mr. Boyd to reproduce some of his experiments by which he arrived at his results before this Society. I consider the doubt expressed by Mr. Flugge-de Smidt, on the work of men well qualified to speak with a good deal of authority, so serious that it is up to Mr. Flugge-de Smidt to bring some proof of his assertions. I am sure that members of this Society will be intensely interested in Mr. Boyd's experiments. If the methods adopted by Drs. Moir and Mavrogordato and the officers of the Mines Department are faulty then I am sure we will all be only too willing to learn from Mr. Boyd. On the other hand we might be able to criticise and find flaws in Mr. Boyd's experiments. In either case difficulties will be cleared up and we will get to know a little more of that wonderful and illusive dust particle. I do think that we should get to know a little more of these experiments as this subject is too serious for generalities.

Mr. C. J. Gray thought people would be getting rather tired of konimeters and dust; but there was one aspect of the matter which he thought had not been fully realised. They had all heard about the differences between sugar tube determinations and the konimeter determinations. Some people were thoroughgoing supporters of the sugar tube. He did not pretend he was himself. He thought it had defects; though it certainly had its good points as well; but, let them take the position as they found it. They found, for instance, that the dust sampling department of the Chamber of Mines using the sugar tube claimed that in Leyner development drives the average result was 2.4 milligrammes per cubic metre. Let them accept that as correct, and that the average diameter of the Leyner particles of dust was 1.5 microns (figures which he understood came from the Chamber were 1.45 and 1.46), though he thought that excessive. It should be possible to convert the sugar tube results into number of particles. One milligramme per cubic metre of air was arithmetically equivalent, for 1.5 micron quartz particles, to approximately 800 particles per cubic centimetre. The 2.4 milligrammes would be arithmetically equivalent to approximately 1,920 particles.

Inspectors of Mines using the konimeter and dark ground illumination obtained, say, a thousand as the approximate average result in these development drives. An exact average count could not be given as inspectors did not count spots over 1,000, but put them down as 1,000+. From those figures it appeared unlikely that inspectors were counting a large number of particles which were not silica.

It might be objected that though the average size of the particles was 1.5 microns it did not necessarily follow that weight could be converted into number by working on that average. That was, of course, the ground for one of the criticisms against the sugar tube; that the sugar tube gave greater importance to the larger particles. But if they took the particular case of the Leyner dust and examined the konimeter slides they would find very few particles over 2 microns in diameter; those who viewed those slides would not think they were making an error more than sufficient to explain the difference by calculating on the basis of 1.5 micron diameter.

The sugar tube results were not absolute proof, but were strong indications that if

inspectors were wrong with the konimeter they erred on the low side.

Mr. R. A. H. Flugge-de Smidt said, with regard to Mr. Gray's last remarks, he would like to point out one or two things. In the course of his (the speaker's) work he had always held that the comparison of the sugar tube with the konimeter was very much more valuable than either separately. In endeavouring to make a comparison he always attempted to judge the character of the particles shown in the spot under the microscope, and, for some considerable period, he used to make an estimate of the average diameter of those particles. He had once published a little article in the *Engineering and Mining Journal* and included some of those tables with columns showing his estimate of the average diameter of the particles. Someone had taken those figures and multiplied them out in order to get the weights, and found they did not compare at all with the gravimetric samples. Let them just take one simple little problem. There were various data giving the number of particles per c.c. which corresponded to 1 milligramme per cubic inch; let them just take the figures roughly in thousands. Sometimes the figures had been given as 2,700, sometimes less. Let them take 2,000; let them take a single count of a thousand particles, and they would say they were all of 1 micron in diameter; that would give them a weight of 0.5 milligrammes. Then, let them take another spot; it was not possible to get them like that, but they would just assume a thousand particles of 1 micron and 10 particles of 10 microns; particles much larger than 10 microns are often included in the weights in the sugar-tube method. The average diameter of these 1,010 particles would be 1.1 microns, only a slight difference as compared with the previous spot, whereas to obtain the equivalent weight per cubic metre one must take:—
 $1,000 + (10 \times 10^3) = 11,000 = 5.5 \text{ Mgs/m}^3$

In comparing those things they got some very interesting results. He had once got a count of 500 particles taken after blowing out the holes in a development end. He endeavoured then to split up those particles into little groups of certain diameters, and he estimated that his sugar tube should give him 30 milligrammes; it gave him 36. The results were quite comparable, provided they got some estimate of the character of those

particles; and he still continued to do it to-day. He took his konimeter spots and then made an estimate of what the sugar tube result should be: the results were sometimes remarkable. He reckoned if he did not get 8 out of 10 correct he was doing badly. That was just by judging the character of the particles.

There being no further discussion, the meeting then terminated.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

RESISTANCE OF CONCRETE TO FIRE.—Professor F. C. Lea and Dr. R. E. Stradling read a paper before Section G of the British Association recently on "The Resistance to Fire of Concrete and Reinforced Concrete." It described a series of experiments carried out in the Department of Civil Engineering of the University of Birmingham, with the object of investigating the effect of high temperature on Portland cement concrete, plain and reinforced. In considering the effect of fire upon reinforced concrete, the problem can be stated under two headings: (1) Is it possible to make concrete which shall retain its strength during and after exposure to the high temperatures liable to occur in a building fire? and (2) Is it possible to prevent the steel from reaching such a temperature that its strength is reduced to or below that required to carry the loads in the building?

In a previous paper one of the two authors has already shown that mild steel at temperatures of little more than 600 deg. C. has a low breaking stress, and that, assuming the concrete itself conduces heat sufficiently to allow the temperature of the steel to rise to more than 600 deg. C., a loaded structure is almost bound to fail. It was also shown that when the surface of concrete is exposed to certain temperatures for periods of from one to four hours the temperature of steel at given distances from the heated surface reached the dangerous point, and that the concrete itself was seriously affected by exposure to high temperature. The details given in the paper were connected chiefly with the first of the two headings mentioned.

Concrete essentially consists of fragments of stone held together by a matrix of mortar, which in itself is an aggregation of fine grains of sand held together by Portland cement. Concrete is very porous. The mode of its failure may be (1) that if the mortar fails, the whole mass will be disintegrated independently of any effect the fire may have on the coarse aggregate; and (2) if all the stones are covered with cement, no increase in total volume can take place unless the cement itself expands.

When expansion takes place stresses will be produced in the covering cement and in the aggregate, unless the expansion of both is exactly equal. These stresses may be sufficient to cause cracking of the cement, or, after cooling, may lead to separation between adjacent materials.

Summarising the data collected, the authors put forward for consideration the following as a short statement of the various factors which effect the failure of cement and sand (quartz) mortars and concretes: The effect of a temperature range of 0 to 98 deg. C. on cement is that it expands at about the same rate as steel, but probably increases in strength; the sand (quartz) also expands, but the relative expansion is not yet sufficient to cause failure. At 100 deg. C. the cement begins to contract, and if held at this temperature, the total contraction is greater than the previous expansion. The volume of the sand probably remains constant. This fact, it is held, is sufficient to cause failure if the sand particles are completely covered and isolated from each other. At 100 to 300 deg. C. the cement is still contracting, the total at 300 deg. being about two and a half times as much as expansion up to 100 deg. The sand, on the other hand, is still expanding. If failure has not already occurred at 100 deg. C., the mortar usually starts to fail during this range. From 300 to 491 deg. C. the cement is still contracting, and it is suggested that the hydrate of lime probably disintegrates during this range. The sand is still expanding. Serious failure usually starts in this range. A temperature of 491 deg. C. is the apparent change point for cement, which commences to expand. From 491 to 575 deg. C. both the cement and sand are still expanding, and 575 deg. C. there is a change of alpha quartz to beta quartz, with a linear expansion of about 0.4 per cent. This usually completes the failure, if not completed before.

Concretes as ordinarily used to-day, containing quartz sand, lose about 20 per cent. of their strength when heated up to a temperature of 550 deg. C. Above this temperature the loss in strength is very much greater, and at about 700 deg. C. the loss is of the order of 70 to 80 per cent. It is suggested that this is probably due to the expansion of the quartz at 575 deg. C., when the alpha-to-beta transformation takes place. Much more fire-resistant concretes can be made by using a fine aggregate made of brick or natural rock (such as basalts or dolerites), which do not contain free quartz in any large quantities. The loss with such a material is only of the order of 30 to 40 per cent. at 700 deg. C., and at even higher temperatures. But although these special concretes may carry their load during a fire, yet the after-effects may be sufficient to cause failure.

There are two other considerations which are of supreme importance in reinforced concrete work, considered as a fire-resisting material. These are the effect of high temperature on reinforced concrete, and the after-effects of fire on concrete. As to the first, with the amount of cover of concrete usually allowed in design, it is somewhat doubtful whether the question of heat conductivity of the concrete is of primary importance. The concrete will spall off the steel reinforcement in many cases long before the temperature of the steel can have risen to

any dangerous degree. In the case of fire tests of large slabs spalling nearly always, if not always, preceded failure. The structure will fail due to the sear being heated to a point at which its strength is seriously affected, but this will usually happen by direct contact with the fire, and not by the heat conducted by the concrete.

It can now be said that small-scale experiments have proved that much money has been wasted in large-scale experiments which were carried out without any real knowledge of the fundamentals of the problem.—PROF. F. C. LEA and DR. R. E. STRAWLING, *Iron and Coal Trades Review*, September 15, 1922, p. 290. (J.A.W.)

BY-PRODUCTS FROM CRUSHING PEANUTS.*—Crushing whole peanuts by the expeller process usually gives a meal containing from 34.4 to 38.6 per cent. of protein. Crushing peanuts from which the hulls have first been removed by the hydraulic process gives a meal containing from 45.3 to 49 per cent. of protein. Peanut meal is an excellent feed. Peanut hulls, however, have a low feeding value and cannot be economically shipped any great distance for use as a feed. It is possible to determine approximately the percentage of hulls in a mixture of peanut and hulls, either from the fibre content or from the protein content. It is probable, however, that the figures obtained from the fibre content will be more nearly accurate. The composition of peanut skins removed by hand differs from that of peanut skins obtained from the peanut-butter plants. The composition of peanut germs is similar to that of the meats, with somewhat lower oil and somewhat higher ash contents. A sample of meats from the Spanish variety of peanuts was found to contain 5 per cent. of sugars and 4.7 per cent. of starch.—J. B. REED, *Journal Franklin Institute*, October, 1922, p. 552. (J.A.W.)

BEHAVIOUR OF REFRACTORIES ON HEATING.—The changes which occur in clays and similar substances on heating are very difficult, and indeed impossible, to follow by purely chemical methods; and the authors have followed others in attempting to derive as much information as possible by a study of physical changes with which chemical changes may be associated.

The methods adopted have consisted essentially in taking the time-temperature curve of a thermo-couple embedded in the specimen and comparing it with the curve given by a second, couple placed in some standard reference substance heated alongside it. If at some temperature the former heating curve lags behind the latter, a change involving an absorption of heat is taking place in the specimen, but if at some point the rate of heating of the specimen is greater than that of the standard substance an exothermic reaction is occurring in the test-piece. The authors have prepared a large number of tables and curves, but they do not at the present stage attempt a complete explanation of the results obtained, which awaits the further evidence they hope to

be forthcoming from other methods of examination which they are applying. The following points emerge, however: (a) Kaolin and all the clays and shales examined show distinct heat absorption at a temperature of 500-530 deg. C. and heat evolution at a higher temperature. (b) This heat evolution occurs at a temperature of 1,060-1,130 deg. C. with pure alumina made by calcining aluminium nitrate at 700 deg. C., but at a temperature lower to an extent of about 100 deg. C. with the bauxites and clays, presumably on account of the impurities associated with the alumina in combination or some form of solution effective for the purpose. (c) Most of the materials show heat absorptions at temperatures below 500 deg. C. presumably due to the loss of water from colloid substances. (d) The phenomena observed with the clays are not primarily dependent on the presence of silica. They are noticeable with alumina and with a red bauxite practically free from silica. (e) For some reason, although alumina calcined at 700 deg. C. and the bauxites and clays show marked heat evolution at about 1,000 deg. C. as noted above, precipitated alumina gave no sign of it. It may be that the change responsible for the heat evolution had occurred with the precipitated alumina before it was heated in the apparatus, but that with aluminium nitrate or silicate the presence of the other material associated with the alumina deferred the change.—H. S. HOULDSWORTH and J. W. COBB, *Iron and Coal Trades Review*, October 6, 1922, p. 498. (J.A.W.)

METALLURGY.

ELECTRIC FURNACE OPERATION.—Undoubtedly the electric furnace from the metallurgical standpoint permits more homogeneity and flexibility in the application of processes and varieties of product than any other steel-making furnace yet devised; the chief characteristic being absolute control of temperature and slag conditions during the melting and finishing period. The flexibility of the electric furnace requires a thorough knowledge of essential principles on the part of the operator, because he must be able to apply any one of them. Lack of adequate knowledge of the operations involved have in many instances led to abuse of the electric furnace, whilst certain mechanical and electrical weaknesses have jeopardised the successful operation of many furnaces. However, properly made electric steel is undoubtedly superior to other grades of steel, close control of the elements producing a product of chemical purity with great density, and good physical properties.

The installation of an electric furnace should not be carried out before a close study of economic conditions such as the cost of electricity, amount of power available, position for freights, costs of labour, etc. In operation, records should be kept of units consumed per ton of steel produced, raw materials, refractories consumed, wear and tear on moulds, and other operating details. Close attention to these items should enable the electric furnace to compete favourably with the open-hearth in

* Issued as U.S. Dept. Agr. Bul. 1906, August 12, 1922.

the ordinary class of steels. In special steels the electric furnace will undoubtedly hold its own, and is more and more replacing the crucible process.

Furnace linings are apt to differ slightly in each works according to their practice, but the fundamental principles do not differ; in the case of a basic furnace lining ordinary firebricks, or in some cases magnesite, are used on the bottom, of course the latter being more expensive; but it has been found while minimising the risk of break-outs, magnesite is liable to cause shorting on to the furnace casting. The firebricks are usually taken up to the slag lining, where it is essential that a neutral layer of bricks should be laid, either magnesite or chrome bricks answering for this purpose, and then the usual silica bricks up to the roof. In the case of a bottom-current furnace, usually a mixture of carbon paste and fine chrome is used, this being rammed over the copper conducting plates.

Difficulties are largely avoided if proper attention is given to selecting suitable dolomite, carefully heating, mixing it without using excessive amounts of de-hydrated tar. Ingredients should be carefully rammed in layer after layer, care being taken that no soft patches are overlooked, as this often leads to trouble during working. The ramming is continued until the bottom and banks are complete. In the case of an acid furnace bottom this may be formed by using peanut size ganister mixed with a little tar to bind; this should be rammed and burnt in the usual manner and fettled with sand, etc. Great care is required when bricking the roof of an electric furnace

will be found more economical in the long run by not retarding the working, as is usually the case when using dirty scrap which causes undue accumulations of slag, loss of metal, and over-oxidised metal. It is usual to charge 25 per cent. of heavy scrap either in the form of shearings, punchings, or heavy turnings; also it will be found more economical when charging to distribute the light turnings round the lining, this serving to protect it during the melting period. Lime, and if necessary, a little oxide, is charged in with the scrap. After melting, and assuming that the analysis is correct and the metal sufficiently hot, the slagging operations are begun, which entails the entire removal of the oxidising slag.

This operation entails two distinct operations: (1) Oxidation of the elements silicon, manganese, carbon and phosphorus. (2) After making the second slag, the reduction of sulphur and any oxides left from the preceding operation. After slagging, the necessary amount of carbon is added either in the form of anthracite or coal duff, which is readily taken up by the metal. It is usual to add a little ferro-silicon, fluor-spar, lime, etc., to form the necessary reducing slag. The current is again switched on at a lower voltage, and preparations made to get the slag fluxed into proper condition for de-oxidising and desulphurising the metal; this condition is only obtained by careful observation of the temperature and slag conditions. It is essential that a good thick fluid slag is maintained by adding lime and fine carbon; if the slag should thicken up, a little spar will be found to be beneficial. Modern practice in killing the metal is to use

Typical Melting and Finishing Slags of a 7-ton Heroult Furnace.

Cast. No.	Slag Sample.	SiO ₂ .	FeO.	Fe ₂ O ₃ .	CaO.	MgO.	MnO.	P ₂ O ₅ .	S.
1	Melting	9.40	21.02	8.80	38.80	10.52	3.55	1.65	0.42
1	Tapping	30.40	0.68	—	64.00	3.07	0.22	—	0.42
2	Melting	11.70	23.04	8.00	37.40	6.99	4.70	2.41	—
2	Tapping	21.60	2.04	—	64.60	3.87	1.20	—	0.10
3	Melting	10.50	19.58	6.40	44.40	8.00	4.00	2.75	—
3	Tapping	11.40	—	—	—	—	—	—	0.27
4	Melting	13.20	13.00	5.20	48.10	7.29	4.30	1.39	—
4	Tapping	9.60	1.01	—	66.20	3.00	1.48	—	0.91
5	Melting	10.20	21.31	9.60	41.60	6.70	3.16	2.32	—
5	Tapping	13.30	1.57	—	63.00	—	1.30	—	0.80

owing to the weakening effect of the electrode holes. Due allowance for expansion must be made when using silica bricks. In some cases bauxite bricks have been used with remarkable success, but allowance must be made for contracting by keying them tightly together. Pre-heating should allow for the complete burning in of the bottom, due care being taken to avoid chipping of the brickwork through neglecting the temperature. In general practice it is found that a constant low temperature is preferable, this ensuring a good burning-in without risks, allowance as to period of time required being made according to the size of the furnace.

FURNACE OPERATION.

Most kinds of scrap can be used, but if possible only the best should be employed, as this

the least possible amount of silicon; this is only possible if the right temperature is observed; nothing causes wild metal more than erratic temperatures, which disturb the slag conditions. Prior to tapping, the necessary alloys, if any, are added.

The melting slags as referred to in the table were taken just before slagging, whilst the tapping slags were obtained just before tapping. It will be noticed that a few of the tapping slags are low in SiO₂. In these particular casts a little calcium-carbide was used to help the slag, and was found most beneficial; it appears to have a retarding effect on the loss of silicon, and in each cast it was found that the theoretical addition was more or less retained in the steel.—WALTER SCOTT, *The Iron and Coal Trades Review*, October 13, 1922, p. 538. (J.A.W.)

TESTING COKE.—In the course of a paper on "Coke Requirements of the Iron Trades," read before the Lancashire Branch of the Institution of British Foundrymen at Manchester, Mr. A. W. Young (assistant engineer in the Burnley Gas Department) remarks that the crushing strength of coke is not a satisfactory test for its hardness when judged from the ironmaker's point of view, as he requires a coke which has not only a hard cell structure, but a coke which, although it may break up from the large lump to smaller pieces, will not crush down to smaller stuff and cause blocking of a cupola. At the present time there is no universally accepted method of mechanically testing coke, but certain tests have been proposed by interested parties in this country. Mr. G. D. Cochrane, Mr. G. Jones and Mr. G. W. Henson have each invented machines for testing coke hardness. The machine used by Mr. Cochrane, designed for him by Mr. Jones, consists of a drum with two angle bars fixed inside to turn the coke sample over. The sample to be tested weighs 56lbs. in lumps up to the size of a man's fist. The sample is dried on hot plates for twenty-four hours, and 28lbs. of the dried coke is placed in the drum, which is given 1,000 revolutions at the rate of 18 revolutions per minute. The sample is then sieved over a half-inch mesh, and the quantity remaining on the sieve is stated as a percentage of the original 28lbs., and is called the hardness of the coke. Mr. Cochrane found as a result of his tests that the output of his blast furnaces fell as the "coke hardness" fell. When the hardness was between 74 and 76 he had very little trouble, but when the hardness fell below 74, the output of the plant fell considerably. The "shatter test" has received a large amount of favour in the United States, being adopted by the Bureau of Mines, and is used to determine the relative breakage of the coke. It consists, usually, of dropping a known weight of coke on to an iron plate from a height of six feet. The apparatus consists of a box having a capacity of 100lbs. of coke, mounted on supports, the cast-iron shatter plate being six feet below the bottom of the coke box. Boards about eight inches in height protect the edges of the cast-iron plate against loss of coke. The bottom of the coke box is provided with hinged doors which are held closely by a latch and are arranged to swing open freely, when the latch is released, thus permitting a free fall for the coke. A sample of approximately 50lbs. of lump coke is placed in the box, and the sample is dropped four times on to the cast-iron plate, the small material and dust being returned to the box with the large coke after each drop. After the coke has been dropped on to the plate four times, it is screened over a two-inch mesh screen, the coke remaining on the screen and that passing through being weighed separately, thus determining the amount of breakage. Carrick Anderson gave the following specification in terms of maximum percentages to be fulfilled by a coke employed for iron smelting: Water, 1 per cent.; ash, 9 per cent.; dust on delivery, 6 per cent.; phosphorus, 0.018 per cent.; and sulphur, 1 per cent. In addition there should be for blast-furnace coke 40 to 50 per cent. pore space, or for foundry coke, 25 to 40 per cent., and hardness of 1,140lbs. per

sq. in., or 80 kg. per sq. cm. The weight of 1 cub. cm. of the coke (dried at 100 deg. C.) should be 0.8 to 0.9 grs. (equal to from 13 to 14.75 grs. per cub. in.). Thaler, in 1915, made an investigation into the valuation of coke, and remarked that cokes for foundry and blast-furnace use are differentiated in practice, but there is very little difference between them in reality. Coke for foundries should be the largest and best pieces, and as a rule should contain less moisture and sulphur.—A. W. Young, *Iron and Coal Trades Review*, December 1, 1922, p. 811. (J.A.W.)

MISCELLANEOUS.

CONTAMINATION OF AIR WITH CARBON MONOXIDE.—Yandell Henderson and Howard W. Haggard, of Yale University (*Jour. Ind. Eng. Chem.*, 1922, xiv., 229-236), draw the following conclusions from their experiments in which the period of exposure was several hours. The time of exposure in hours is multiplied by the concentration of the carbon monoxide in parts per 10,000 of air. If the product equal 3, no perceptible physiological effect is noted; if it equal 6, the effect is just perceptible; if it equal 9, headache and nausea are produced; if it equal or exceed 15, conditions are dangerous to life. Exercise and physical work increase the rate of absorption of carbon monoxide from a contaminated atmosphere. Upon return to fresh air, the carbon monoxide which has been absorbed by the blood, is excreted through the lungs; from 30 to 60 per cent. of the absorbed carbon monoxide is excreted hourly. Carbon monoxide is the only toxic constituent of importance in the exhaust gas from gasoline. Illuminating gas and the exhaust gas from coal tar distillate contain other toxic constituents as well as carbon monoxide.—Y. HENDERSON and H. W. HAGGARD, *Journ. Franklin Inst.*, July, 1922, p. 108. (J.A.W.)

CHANGES OF ADDRESS.

- BARTON, B. V., *1/o* Rhodesia Broken Hill Development Co., Ltd.: 33, Glenferrie Road, Malvern, Victoria, Australia.
 COUPERTHWAIT, J., *1/o* Brakpan Mines, Ltd.: Springs Mines, Ltd., P.O. Box 54, Springs, Transvaal.
 DIERING, J. H. A., *1/o* Vaalhoek Mine, Pilgrim's Rest: Glyn's Lydenburg, Ltd., Sabie, Transvaal.
 JOHNSON, J. H., *1/o* Aurora West G. M. Co., Ltd., Maraisburg: 85, Pandora Road, Kensington East, via Denver Post, Johannesburg.
 LLOYD, J. J., *1/o* Brakpan: Half-Way Hotel, P.O. Koosfontein, via Bloemhof, Transvaal.
 OLSEN, A. C. L., *1/o* Modderfontein East, Ltd.: Geldenhuis Deep, Ltd., P.O. Box 54, Cleveland, Johannesburg.
 PROBERTS, T., *1/o* Village Main Reef G. M. Co., Ltd.: Ferreira Deep, Ltd., P.O. Box 1021, Johannesburg.
 RICHARDSON, J., *1/o* Witwatersrand G. M. Co., Ltd.: New State Areas, Ltd., P.O. Box 41, Springs, Transvaal.
 YATES, Prof. J., *1/o* Parktown North, Johannesburg: White Lodge, Jan Smuts Avenue, Sachsenwald, Johannesburg.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA

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Vol. XXIII.

MARCH, 1923.

No. 9.

Proceedings

AT

**Ordinary General Meeting,
17th March, 1923.**

The Ordinary General Meeting of the Society was held in the Assembly Hall, Scientific and Technical Club, 100, Fox Street, Johannesburg, on Saturday, the 17th March, 1923, at 8 p.m., Prof. G. A. Watermeyer (President) in the chair. There were also present:—

23 Members: Messrs. H. R. Adam, C. J. Gray, F. W. Watson, J. R. Thurlow, Andrew King, John Watson, H. A. White, J. L. Willey, J. A. Woodburn (Members of Council), H. D. Bell, J. A. Boyd, J. M. Dixon, H. Goodwin, E. Homersham, J. V. Muller, F. D. Phillips, J. F. Pyles, C. B. Simpson, R. A. H. Flugge-de Smidt, S. de Smidt, H. R. S. Wilkes, L. B. Wilmot and A. A. Wood.

4 Associates: G. J. V. Clarence, R. W. Irwin, R. Kethel and H. Rusden.

3 Visitors and H. A. G. Jeffreys (Secretary).

MINUTES.

The President regretted that the *Journal* containing the minutes of the proceedings of the Special General Meeting held on the 10th February, and the Ordinary General Meeting held on the 17th February, 1923, had not arrived yet. It was expected it would be in the hands of members early next week. Under the circumstances he would ask members to postpone confirmation of the Minutes until the next meeting.

NEW MEMBERS.

A ballot was taken for the election of new members, and the following gentlemen were declared unanimously elected:—

GOODWIN, HARRY, Modder Deep Levels, Ltd., P.O. Box 326, Benoni, Cyanider.

MACKENZIE, WM. A., New Consolidated Gold Fields, Ltd., P.O. Box 1167, Johannesburg, Company Manager.

GENERAL BUSINESS.

BRITISH EMPIRE EXHIBITION.

The President read the following notice with regard to the British Empire Exhibition:—

BRITISH EMPIRE EXHIBITION, 1924.

It was announced some time ago that the Union Government has decided to participate in the British Empire Exhibition, and that a sum of £50,000 has been voted for that purpose. The British Empire Exhibition Committee (Union) has addressed the Associated Scientific and Technical Societies of South Africa on the subject, inviting the co-operation of the Constituent Societies in assisting South Africa's Exhibit.

Suggestions and ideas from members are invited, and these may be communicated to the Secretary of this Society, who will also be glad to hear of any interesting exhibits to be either loaned or donated.

The Constituent Societies have also been asked to arrange for papers to be read during the Exhibition through their Associated Societies in London.

Members are invited to submit to the Council original papers dealing preferably with either the resources of this country, special research or novel practice.

It is also proposed to have a pamphlet prepared setting forth the activities of the Associated Scientific and Technical Societies and the Constituent Societies which could be distributed at the Exhibition.

Mr. A. H. Tatlow, the Organising Secretary of the Exhibition Committee (Union) will address the members of the Transvaal Manufacturers' Association on the subject in this hall on Tuesday, the 27th inst., when

members will have an opportunity of hearing further details.

Mr. J. R. Thurlow asked whether in connection with the papers which had been asked for it meant they would be presented in conjunction with the Institute of Mining and Metallurgy.

The President replied in the affirmative and added that the I.M.M. were associated with them (the Society) and that some London Society would present these papers if they were forthcoming.

Mr. J. R. Thurlow asked whether, in that case, the papers would remain the exclusive property of the parent Society through whom they went.

The President said that was a matter which they had not considered in Council, but he had no doubt they would be published in the Society's *Journal*.

Mr. J. R. Thurlow thought it was rather an important point, and that perhaps the Council would obtain some information from Mr. Tatlow on the matter.

HONORARY LIFE MEMBER: DR. J. MOIR.

The President said it gave him great pleasure to announce that in recognition of the valuable services and support rendered to the Society by Dr. Moir, Past President, the Council had conferred upon him the Honorary Life Membership of the Society. (Applause.)

THE SOLUBILITY OF GOLD AMALGAM IN CYANIDE SOLUTIONS.

By H. A. WHITE.

For various reasons the trend of current practice on the Witwatersrand is to eliminate the use of mercury in the direct recovery of gold. Where this has been done in the case of some of the mines of the Eastern Rand a tendency to slightly lower residues, especially for treated sand, has become apparent.

The following experiments were carried out in order to investigate this phenomenon which was not generally anticipated.

A piece of pure gold was rolled out on the bullion rolls as thin as convenient and from it two pieces as nearly as possible 2.7×1.7 cm. each were cut and annealed.

Piece A weighed 498.9 mgm.

Piece B weighed 499.1 mgm.

The thickness was therefore approximately 56 microns in each case. Piece B was dipped rapidly into pure mercury and at once wiped as free from mercury as possible. The weight was then 515.4 mgm., indicating that 16.3 mgm. of mercury was retained firmly on the 9.18 sq. cm. of exposed gold surface. This would be 1.776 mgm. per sq. cm., equivalent to 0.05 oz. per sq. foot of gold surface, and would indicate that the thickness of the mercury film was only 1.3 microns.

A preliminary experiment was made by hanging up by means of cotton thread each piece in 500 cc. of a cyanide solution (KCN 0.027%, NaIO 0.006%) saturated with oxygen at 28° C. and three inches below the level of the solution which was contained in glass jars in the shade.

After 24 hours the pieces of gold were removed and weighed and the solutions assayed.

The pure gold lost 9.5 mgm. or 1.3 mgm. per sq. cm. per 24 hours, while the amalgam lost 4.3 mgm. of gold (0.5 mgm. per sq. cm. per 24 hours) and mercury by difference 5.1 mgm. or 0.55 mgm. per sq. cm. per 24 hours.

The protective alkali being too low is no doubt responsible for the low rate of solubility in both cases, but it is clearly indicated that mercury reduces the rate of gold solution to one-half. In this experiment the rate of solution of the mercury incidentally supports the deduction made in 1919 (see *Journal*, Vol. XX., p. 1) that metals which are soluble in cyanide solutions have their relative rates of solution proportional to the chemical equivalents.

$$\begin{aligned} \text{Thus } 2\text{Au/Hg} &= 394.4/200.6 = 1.9 \\ &\text{and } 9.5/5.1 = 1.8 \end{aligned}$$

This indicates that a mercury film of this thickness is attacked in much the same way as pure mercury while the limit to the rate of gold solution is here the speed with which it can diffuse through the thin layer of mercury. The same pieces of gold were used for the main experiment, which was carried out in exactly the same manner for five days, with one complete change of solution half-way in order to keep up the strength of the solution in cyanide, oxygen and alkalinity. The solution in each case was saturated with oxygen at 19° C. and the pieces were hung one inch below the level of the solution, which assayed 0.027% KCN and 0.025 NaHO (practically optimum strength). The pieces of gold were then washed, dried and weighed, and the total solution assayed for gold in each case.

The pure gold lost 166.6 mgm. or 3.63 mgm. per sq. cm. per 24 hours, while the amalgam lost 77.5 mgm. of gold, equal to 1.69 mgm. per sq. cm. per 24 hours, and 7.5 mgm. of mercury or 0.16 mgm. per sq. cm. per 24 hours.

The above deduction that gold will only dissolve at half the usual rate when the surface is covered with mercury is therefore confirmed, and this suggests the solution of the problem of the lower residues in the absence of mercury. That there is ample mercury present in the pulp usually leaving our crushing plants to coat all the gold present, the following calculations will fully demonstrate. Assuming the loss of mercury to be 0.1 oz. per ton milled with a tailing assaying 5 dwt. of gold per ton, and that the average gold particle is one four-hundredth of an inch or 60 microns in diameter with one face exposed, it may be calculated that the total surface is 78 sq. cm., so that the mercury if evenly spread would form a film of 28 microns thick or more than 20 times as much is present as the first experiment found to be necessary.

If the fact be remembered that crushing is usually done in a closed tube mill circuit and a hint be taken from ordinary flotation experience with gold and a tar coating, it will be sufficiently obvious that the chance of a gold particle in the mill pulp being completely covered with mercury is very high. It is of course generally understood that it is the most unfavourably situated particles of gold which determine the length of treatment profitable and the residue value obtained, and it may therefore be confidently antici-

pated that where amalgamation is totally dispensed with and no particles of coarse gold can escape the final cones, either a somewhat lower residue will be obtained or an increased tonnage can be put through the same plant. Before leaving the subject it may be interesting to note that the experiments above showed that the coated gold still appeared to be completely amalgamated though the film of mercury had been finally reduced to 0.3 micron average thickness. Under the microscope a pattern of central holes surrounded by one or two circular rims, like craters, was evident, and the deeper parts showed the yellow tint of exposed gold which possibly protected the mercury from further solution. It may also be noted in passing that the dissolved mercury in our working solutions is of benefit in removing alkaline sulphides, etc., but that any small advantage lost in this way can readily and cheaply be regained by the use of lead salts.

Mr. J. R. Thurlow: proposed a hearty vote of thanks to Mr. White for his very useful paper, and while he thought members would agree it would be premature to discuss it that night, he felt quite sure they would have something to say about it later on. As usual, Mr. White had given them something which would make them think. The paper was quite up to his usual high standard, and commanded their appreciation. The logic advanced and the correlation of that logic to the everyday work of our members would furnish the inspiration for some good papers and discussions on the subject.

OBSERVATIONS ON THE TIN-ANTIMONY-LEAD ALLOYS.

By R. W. IRWIN.

That the importance of the physical and mechanical properties of anti-friction metals and kindred alloys is not appreciated to anything like the extent that they should be is evident from the scanty literature on this subject.

Dr. Rosenheim draws attention to this in a report of the researches of the National Physical Laboratory, 1921.

The significance of this fact was brought very prominently to the notice of scientists when during the late war a bearing metal suitable for the very high speed aeroplane engines had to be devised.

The importance of the structure on the properties of alloys in service, is appreciated

by practical engineers, and is therefore no longer of purely theoretical importance. Mr. George Hughes, Chief Mechanical Engineer to the Lancashire and Yorkshire Railways says: "The success of a bearing metal alloy depends more upon structure than absolute composition, the proper distribution of a number of the tin-antimony crystals in a softer matrix of tin or lead being essential."

The important point to realise is that most anti-friction alloys have quite complex structures, and consist of metals, solid solutions of metals, eutectics, and chemical compounds, and that the skill employed in the manufacture, determines the amount and the state of these various constituents and makes

the alloy suitable or otherwise for its specific purpose.

The essential feature of an antifriction metal is that it should consist of two compounds, the one hard and the other comparatively soft. The hard constituent usually consists of a chemical compound and is set in a matrix of the softer constituent which is usually a eutectic, a solid solution, or a mechanical mixture of the two. The hard constituent takes the weight of the journal and has a low co-efficient of friction, while the softer constituent is sufficiently plastic to allow the bearing to conform to any irregularities of alignment, and thus obviate excessive local friction.

A practical illustration of the importance of structure and heat treatment will perhaps prove of interest. Mr. Archbutt quotes an example in connection with the manufacture of slide valves on the Midland Railway. These are made of bronze containing 15% of tin and are tested by putting them under a falling weight of 112lbs. with a blunt knife edge. The valves are cast in sand, and when supported on a nine inch centre, a single blow from a height of eight feet is sufficient in most cases to crack them, while three blows from the same height will break them in half. If, however, the castings are quenched from a temperature just above 500° C., they will stand four blows before cracking and about nine blows before breaking. In some cases fifteen blows from a height of eight feet, and three from a height of ten feet were required to break them. Owing to their unusual strength and toughness, the practice of quenching the castings was adopted, but soon had to be abandoned owing to the rapid wear of the valves. The explanation is simple. The cast alloy consists of particles of a hard constituent embedded in a soft matrix, a characteristic and essential structure of all alloys required to resist friction or abrasion. The quenched alloy on the other hand possesses a homogeneous structure, which is ill-adapted to resist wear.

Having briefly outlined some of the more important points in connection with antifriction alloys, in general, it is now proposed to pass on to some investigations which have been carried out on alloys.

The important point that it is desired to emphasise is the very marked effect which the mould temperature has on the lead base alloys. This was determined some time ago whilst engaged in an investigation on the properties of antifriction alloys (see *J.S.A.I.E.*, Dec., 1922).

If the metals were poured into moulds at room temperature they were hard and stood up well to compression, but were somewhat brittle, whereas, if poured at 150° C. they were soft and plastic, and compressed rapidly. On examining these metals under the microscope it was evident that the flattening was due to the compression of the matrix, and not to a breaking down of the hard crystals. An investigation, therefore, was carried out on some of the more important alloys entering into the composition of the matrix of the white metals, and the results are perhaps of sufficient interest to bring to the notice of this Society.

The testing instrument used was the drop hammer, as described by Mr. T. B. Lynch of the Westinghouse Electric and Manufacturing Co., in the Proceedings of the American Society for Testing Materials, Vol. XIII., which he claims show results that indicate the qualities of babbit more clearly and definitely than any other quick test. The test specimen, if soft, preens out, if brittle it breaks, while if soft and brittle it preens out and crumbles. Experience with this instrument confirms the above, and it is a splendid indicator of the qualities of an antifriction metal, whereas friction testing machines have failed to give reliable results, and the Brinell hardness number alone is not sufficient, as it fails to differentiate between toughness and brittleness.

Miss Fry and Dr. Rosenheim (*Journal of the Institute of Metals*, No. 2, 1919, Vol. XXII.) have carried out a series of experiments on a tin base metal which was heated to and poured into moulds at varying temperatures, and the hardness of the metal was then determined. It was found that the Brinell hardness number variations were comparatively small, the lowest number being 25.2, and the highest 32.7. The hammer test on the tin base metals bears this out, as the compression under the hammer is not greatly influenced, either by the temperature of pouring or by the temperature of the moulds, and their rate of cooling. It is very different, however, with the lead base metals as here the pouring and mould temperatures have a very great influence on the properties of the alloy, as revealed by the hammer test, and it is proposed to give you the results of some of the experiments which will be illustrated by means of the microscope and actual test specimens.

Cylinders of these alloys measuring $\frac{1}{8}$ " \times $\frac{5}{8}$ " diameter were subjected to the number of blows indicated in each series of experiments

and under similar conditions. The hammer used delivers a blow of about 2ft. pounds, and the compressions were measured by means of a micrometer screw gauge. It was necessary to select a mould temperature sufficiently low to allow all the alloys to be tested under similar conditions. As the eutectic of the Sn Pb series solidifies at 180° C., 150° C. was selected as a suitable mould temperature for the one series of experiments, and the ordinary room temperature for the other. The mould used for the casting of the specimens took about 2½ hours to cool from 150° C. to room temperature.

A typical lead base metal (1) of the composition Pb 70 Sn 15 Sb 14 Cu 1, and a typical tin base metal (2) of the composition Sn 84 Sb 10 Cu 6, were selected and compared under the hammer as follows:—

Metal.	No. of Blows.	Temp. of pouring metal.	Temp. of mould.	Compression.
(1)	3,850	350 approx.	20° C.	0.050"
	3,850	350 approx.	150° C.	0.209"
(2)	3,850	350 approx.	20° C.	0.120"
	3,850	350 approx.	150° C.	0.132"

The extreme sensitiveness of the lead base metal for such a narrow range of temperature is remarkable, whereas the influence of the temperature on the tin base metal is very slight in comparison.

The compression of the lead base metal cast at 20° C. is less than half that of the tin base metal cast at the same temperature, whereas when cast at 150° C. the compression of the lead base metal is about 1½ times greater than that of the tin base metal cast at this temperature.

We note that the structure of these alloys is very different under the microscope. (1) consists of antimony tin crystals (Sb Sn) set in a matrix of a mechanical mixture of lead and antimony tin; (2) consists of tin antimony (Sb Sn) and tin copper crystals (Sn Cu₃) set in a matrix of the tin antimony eutectic.

Specimens were prepared of the various alloys which enter into the constitution of the two above metals and were compared under the hammer. The two primary metals, lead and tin, were first taken.

Metal.	No. of Blows.	Temp. of Mould.	Compression.
Lead (commercial)	50	20° C.	0.327"
	50	150° C.	0.354"
Tin (commercial)	50	20° C.	0.110"
	50	150° C.	0.125"

Both these metals compress slightly more when poured into a mould at 150° C. than

when the mould is at 20° C. This is due to the crystals of the metal being larger when poured into the mould at the higher temperature and slowly cooled. The difference in the degree of compression of the two metals, however, is very marked, the compression of the lead being three times that of the tin. Cylinders of a solid solution of Sn in Pb and of Sb in Sn were next prepared and compared under the hammer.

As Rosenheim and Tucker were able to get 16% of Sn into solution in Pb by long annealing; an alloy of 16% Sn was selected, but as is seen by the microphotograph all the Sn is not in solution.

Stead found that up to 7.5% Sb alloys consist of a solid solution, but above this point cubes of a compound Sb Sn separated out when the alloy froze. A mixture containing 7% Sb was therefore selected as a suitable alloy, but crystals can be noticed in the microphotograph owing to equilibrium not having been attained.

Solid Solution.	No. of Blows.	Temp. of pouring metal.	Temp. of Mould.	Compression.
		approx.		
16% Sn, 84% Pb	200	350	20° C.	0.280"
	100	350	150° C.	0.238"
7% Sb, 93% Sn	2,000	350	20° C.	0.194"
	2,000	350	150° C.	0.192"

Note the effect of the temperature of the mould when the Pb is the predominating metal, and the little if any effect it has when the Sn predominates. Both these solid solutions stand up to compression very much better than the single metals did.

From the microphotographs of both these solid solutions it is noticed that when the metals were poured into moulds at 20° C. there is more of the hard constituent observable, which shows a greater separation of this constituent, and this is probably the cause of the greater resistance to compression of these solid solutions, when poured into moulds at 20° C., as compared with the samples poured at 150° C. Cylinders of the eutectics of Sb and Pb, and Sn and Pb were then examined.

Eutectic.	No. of Blows.	Temp. of pouring metal.	Temp. of Mould.	Compression.
		approx.		
13% Sb, 87% Pb	2000	350	20° C.	0.171"
	2000	350	150° C.	0.246"
		rapidly cooled		
	500	350	150° C.	0.275"
63% Sn, 37% Pb	500	280	20° C.	0.235"
	500	280	150° C.	0.231"

The sensitiveness of the Sb Pb eutectic to the mould temperature is very marked. The eutectic poured at 150° C. and rapidly cooled by being plunged into water stands up to compression much better than when slowly cooled. This indicates a structural arrangement in the rapidly cooled eutectic, which is much better able to withstand compression than is that of the slowly cooled alloy. The eutectic alloy is the result, so far as its structure is concerned, of the simultaneous crystallisation or freezing of the two components, and its crystallisation is similar to that of a pure metal, in so far as solidification is completed at a constant temperature.

As a rule, one of the metals comprising the eutectic acts as the predominant partner, and its own crystalline form or habit determines the way in which the whole structure is arranged. Both pure metals and eutectic consist of juxtaposed crystals, the difference being that the crystals of the predominant partner are merely skeletons, and the interstices are filled in by the other metal. From the microphotographs of both the Sn Pb and the Pb Sb eutectics it appears that the crystallisation of the predominant partner is more marked in the rapidly cooled eutectics. This crystallisation is very much more exemplified in the case of the Pb Sb eutectic than in that of the Sb Sn eutectic; and this accounts for the marked difference in the compressibility of the Pb Sb eutectic when quickly or slowly cooled. The ideal structure to withstand compression without being brittle is to have a hard constituent embedded in a softer and more plastic one. We obtain this structure in the quickly cooled Pb Sb eutectic, but not in the slowly cooled one. We thus realise how important it is to regulate the mould temperature when pouring the Pb Sb eutectic or an alloy containing this eutectic, as by exercising due control of temperature we can either produce a hard alloy well able to withstand compression or a soft and ductile alloy. This property is not only a function of the Pb Sb eutectic but seems to be common to the alloys of a Pb base, as we saw the same effect in the solid solution of Sn in Pb, and we also observe it in the following two ternary alloys.

- (1) 40% Pb, 57.5% Sn, 2.5% Sb, which is a mechanical mixture of Pb and Sn *a*,
- (2) 37.5% Pb, 60% Sn, 2.5% Sb, which consists of white dendrites of Sn and surrounded by binary Pb Sn *a*.

	No. of Alloy. Blows.	Temp. of pouring metal.	Temp. of Mould.	Com-pression.
(1)	2,000	280°C. approx.	20°C.	0.241"
	1,000	280°C. approx.	150°C.	0.259"
(2)	2,000	280°C. approx.	20°C.	0.279"
	1,000	280°C. approx.	150°C.	0.291"

These ternary alloys in common with the binary alloys above have a more homogeneous structure when slowly cooled than when rapidly cooled, which explains their difference in compressibility. These two alloys were selected as they lie one to either side of a transition point in the Sn-Sb-Pb diagram, and it is interesting to note the difference to the amount of compression which the small alteration in composition has made, the alloy having the higher percentage of tin being the more compressible.

From the above experiments it is obvious that some alloys in daily use are very sensitive to the treatment which they receive and require great care in their handling if the best results are to be obtained.

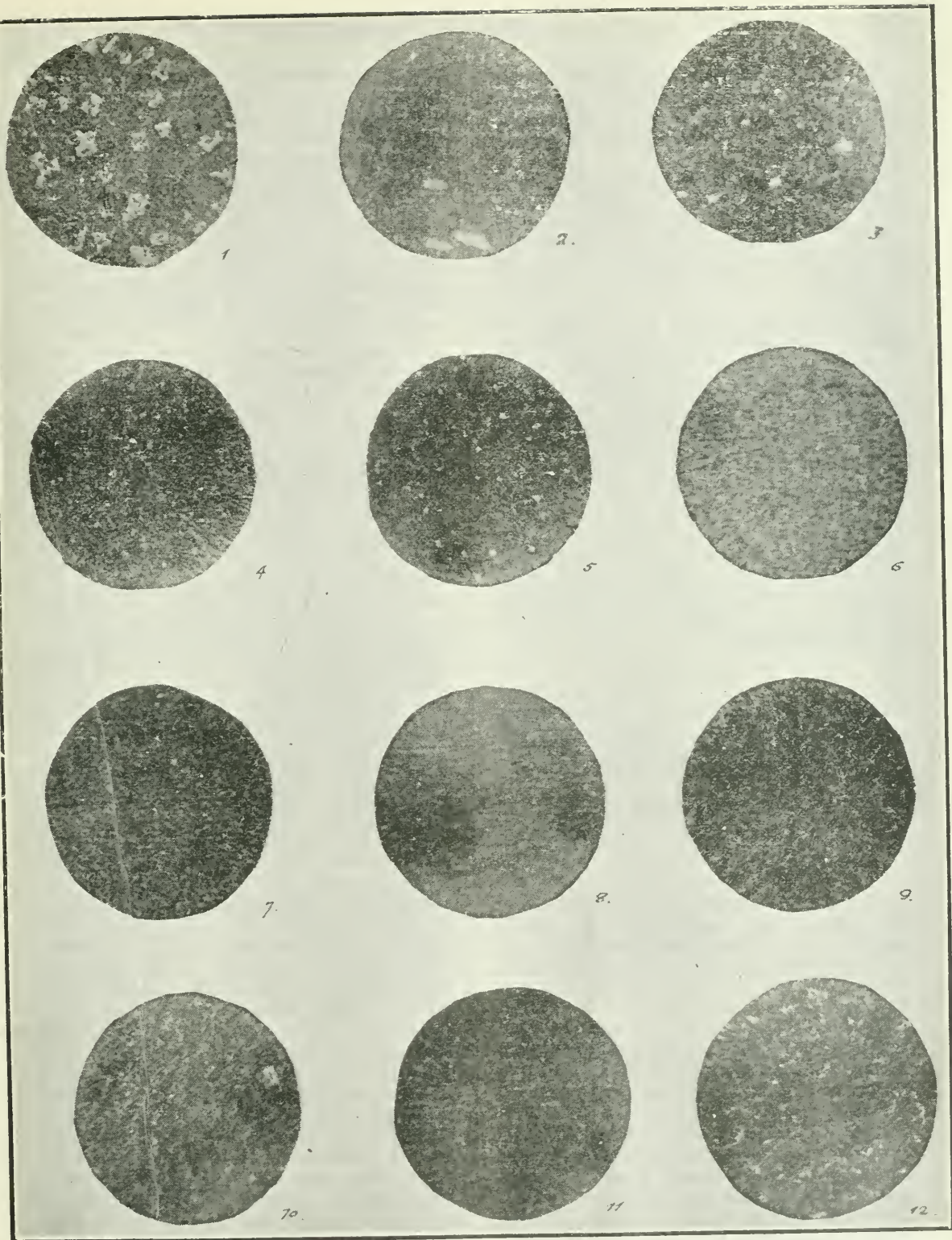
There is a wide field here waiting closer research and it is to be hoped that this cursory investigation may be instrumental in inducing someone to carry the research further.

In conclusion, I wish to thank the management of the Germiston Laboratory and Industrial Co. for the use of their apparatus in carrying out this work.

Mr. F. W. Watson proposed a very hearty vote of thanks to Mr. Irwin for his interesting paper. The paper was extremely welcome, as, to the best of his knowledge, it was the author's first contribution to the Society, and had involved a lot of real work.

There was a time when engineers relied too much on the composition of the metal without giving due regard to the structure. Some time ago the Chamber of Mines drew up a table of six standard metals ranging from No. 1 containing 84% of tin, 10% of antimony and 6% of copper with no lead down to No. 6, which had 80% of lead and no copper. Naturally, on account of cheapness, the last metal was used whenever possible, and the author's data on the effect of mould temperature on lead base metals should be of considerable value.

He was very interested to note that the experience of engineers on the Home railways had been quoted in the paper, and thought that it might be possible to obtain some discussion from the authorities on the South African Railways. He suggested that the Society communicate with the railway engineers with a view to obtaining a contribution.



1. Lead Base Metal. 2. Solid Solution of Sn in Pb 150° C. 3. Solid Solution of Sn in Pb 20° C. 4. Solid Solution of Sb in Sn 20° C. 5. Solid Solution of Sb in Sn 150° C. 6. Sb-Pb Eutectic 20° C. 7. Sb-Pb Eutectic 150° C. 8. Sb-Pb Eutectic 150° C. 9. Sn-Pb Eutectic 20° C. 10. Sn-Pb Eutectic 150° C. 11. 37.5 p.c. Pb, 60 p.c. Sn, 2.5 p.c. Sb, 150° C. 12. 40 p.c. Pb, 57.5 p.c. Sn, 2.5 p.c. Sb, 150° C.

Mr. H. R. Adam, in seconding the vote of thanks, said he was glad Mr. Irwin had given them this paper. The Society had too few papers on this important branch of metallurgy.

He was not prepared to offer any critical remarks that evening; the paper was one that required to be read beforehand. He thought there was probably a considerable amount of controversial matter in it and would perhaps find something more to say later on.

On one point only a remark might be made. The author had made it quite clear that the principle in bearing metals was to have an alloy consisting of a harder constituent embedded in a softer matrix, and on this account he (the speaker) could not understand the purpose of the investigations on eutectic alloys nor could he understand the statements made referring to the freezing out of excess metal in eutectic alloys. If the alloy were of eutectic composition surely there could be no freezing out of excess constituents. However, the author's ideas would no doubt be made clear on reading the paper. He wished to congratulate him on his investigations.

THE APPLICATION OF FLOTATION TO THE ANTIMONIAL GOLD ORES OF THE MURCHISON RANGE.

By H. R. ADAM.

(Printed in *Journal*, July, 1922.)

REPLY TO DISCUSSION.

I have to thank Messrs. Homersham and Simpson for their contributions and also to suggest to Mr. Homersham that he did not read the paper very carefully, otherwise he would not have concluded that all the flotation tests were unsatisfactory. If he will refer to tests (13) and (14) he will find that flotation plays a very important part in the scheme of treatment and that my results in these latter tests are in good agreement with Mr. Simpson's. Of course, the actual recovery by flotation is much lower since "gravity" concentration was introduced and the amalgamation recovery is not so high in test (14) since only the "gravity" concentrate was amalgamated, but the total recovery as amalgam and concentrate was 93%.

The only difference of opinion is that Mr. Homersham thinks that in the case of fresh

sulphide ore about 95% of the stibnite and gold could be recovered as a flotation concentrate, but he gives no figures to substantiate this statement. In test (8) (direct flotation) I found 65 % of the gold in a flotation concentrate amounting to 29% by weight of the total ore treated, and even admitting that the flotation was not carried out under the best conditions, I do not think that by the one operation of flotation a tailing sufficiently low in gold for rejection could be obtained if we confine ourselves to conditions which are feasible in practice.

The gold in these ores is not closely associated with the stibnite. In a previous paper (this *Journal*, June, 1915) I showed that by flotation in hot acid solutions an almost perfect separation of stibnite could be obtained on fairly coarsely ground ore and that the greater part of the gold was left with the tailing. Certainly if the ore is ground much finer and the minerals separation process used a much greater proportion of the gold will be with the froth, but also a much greater proportion of gangue. In addition to the gold attached to gangue particles there is the gold associated with the mineral tetrahedrite, and I found that to separate the particles of this mineral sufficiently for flotation it was necessary to grind very fine. Also in Mr. Simpson's test 95% of the flotation feed passed a 100 mesh screen and the flotation concentrate only contained 83% of the gold. This, of course, was on semi-oxidised material, so it is perhaps hardly a fair comparison.

It is hardly worth while to press the point further: the composition of the ores varies so much that there is bound to be considerable differences in the results obtained. I agree with Mr. Homersham that for a gold recovery of over 90% flotation would be necessary. My suggested treatment scheme which was designed more for the "small worker" consisted of amalgamation "gravity" concentration and flotation, and I might very briefly give the reasons. Amalgamation is included because it gives an immediate return—a rather important point to the "small worker"—and because there exists, near the surface at least, a considerable amount of ore which is fairly free from antimony and is easily amalgamated; "gravity" concentration is introduced because it is cheap, simple, free from patent restrictions and will give a concentrate more suitable for subsequent roasting. The "gravity" concentrate could also

be re-ground with mercury for a further production of amalgam. I have some figures (not given in the paper) which show that an extraction of about 80% of the gold could be obtained by amalgamation and "gravity" concentration, and to bring this to 95% or over it is clear that flotation would be required.

SOME NOTES ON THE EVOLUTION OF THE KONIMETER

BY R. A. H. FLUGGE-DE SMIDT.

(*Printed in Journal, November, 1922.*)

DISCUSSION.

Dr. J. Moir (contributed): I should like to suggest to the author that his review of this subject is more partisan than historical, particularly in regard to his statement that the Kotzé instrument possesses no advantages: such a statement is really astonishing when it is notorious that the Kotzé instrument is the only konimeter which does not leak in practical use.

The author is particularly unfortunate on page 79 in using a formula of Dr. Owens. In lines 15 to 21 we are told that the catch on the second disc is 30% of that on the first, and at the bottom of the same column we are given the first catches as calculated either by the author or by Mr. Boyd, the results being 56, 44 and 52%. Now, anyone who applies Owens' formula will discover that to get the 30% ratio 70% must be caught on the first disc and 21% on the second.

Again, Mr. Boyd is quoted to show that hydrochloric acid is not a success in removing calcium sulphate. It is, however, easy to calculate that the maximum possible amount of CaSO_4 on a konimeter spot is about 0.00002 gram, which would require only half a milligram of HCl to dissolve it. As the discoverer of the HCl process, I strongly object to its being mauled by amateurs, and suggest that someone is using a faulty technique, e.g., exposing to air while taking a microphotograph and thus evaporating the HCl.

Mr. H. R. Adam: I think it will be unfortunate if the discussion on Mr. de Smidt's paper closes without some reference to the question of the illumination used in counting the particles. According to the author the Mines Department uses dark

ground illumination, while for routine work the Chamber of Mines uses transmitted light. Now while the use of dark ground illumination will undoubtedly be of great value in the general study of mine dust it will also, I should think, introduce further complications in as far as the konimeter is used as an instrument for routine quantitative work. Mr. de Smidt makes this clear in his example of a count of about 300 with light ground being 500 with dark ground. It may also be anticipated that the counts with dark ground illumination will vary very considerably according to the degree of perfection of the dark ground condenser system. Thus the count with a simple stop to the Abbé condenser will be quite different to that obtained with a paraboloid condenser, and even confining the method to the former it is difficult to obtain conditions which are clearly defined. The matter may perhaps be summed up in this way: the more perfect the dark ground (i.e., the greater the obliquity of the rays from the condenser and the smaller the proportion of direct rays entering the objective) the greater will be the probability of including harmless material in the counts, until ultimately the difficulty will be to know where to stop counting. The removal of CaSO_4 obviously makes a considerable correction, but it seems to me that we are not justified in using methods of illumination which permit of any great uncertainty as to what is being counted. In any case if comparative results are to be expected the conditions of illumination will have to be defined in detail since even with the simple stop system there is room for wide variation in results.

Mr. Boyd recently gave me the results of a test made comparing the counts of the same spot under different conditions of illumination: he has kindly permitted me to give the figures.

Light ground.	Dark stop.	Dark stop and water.	Paraboloid and oil.	After acid treatment, Dark stop and water.
280	550	750	650	530
260	550	720	600	420
300	550	706	700	460
190	450	450	380	280
250	580	600	480	320
300	480	520	480	320

It is difficult to understand why the figures for the paraboloid condenser should be lower. I should have thought the difficulty would have been to know where to stop counting. According to Mr. Boyd the difficulty of

defining the area to be counted may explain it.

Before concluding, I cannot resist referring to a remark made by Mr. Gray in his contribution to the discussion. Mr. Gray seems to think it a disadvantage that with the circular konimeter as used by the Chamber of Mines the slides should be sent in to be counted by different individuals. If we are seeking for general information on the nature of the dust from different places there may be something in what he says, but, I put it to the chemists here to-night, that in so far as the konimeter is to be used for routine testing, this is exactly what should be done.

Mr. C. J. Gray agreed to a great extent with what Mr. Adam had said on the question of dark ground illumination. It was true that the count obtained with dark ground depended on the method of illumination. The count could be increased in most slides by using more perfect condensers and stronger illumination. The Mines Department used certain condensers and illumination which falls far short of showing ultra-microscopic particles. The Chamber of Mines had obtained a parabolic condenser, which, however, was not used in routine work, with which counts could be obtained very considerably in excess of those which the Department got for the same slides. Whether the Chamber with that condenser approached the ultra-microscopic he could not say.

That was the difficulty with regard to dark ground illumination. For comparable results they must have equal illumination, but with any ordinary illumination they could not say they were counting all the particles which were present. If for instance the count was 1,000 there might be 1,500 or 2,000 particles present: they could say only that they were certain there were at least 1,000.

Now with light ground illumination the position was that if that illumination were excessive, they could not see the particles, because the particles of quartz were lost in the flood of light. If the illumination were reduced in degree, they might count, say, 300 of the coarser particles in a spot giving a dark ground count of 1,000. If it were reduced further, they might get the point of maximum visibility with regard to light ground, and bring up the count to the same as the dark ground count with Mines Department methods. If it were reduced still

further, particles would be lost in the darkness. He had taken a slide himself; chosen three particles which he could recognise; counted the number of particles in the triangle formed by joining those three particles by imaginary lines; and by very carefully adjusting the illumination, had got approximately the same counts with different magnifications and with both light ground and dark ground. In ordinary work with light ground, one would not get the same result: it required too much care in adjustment of lighting and in counting particles on the limit of visibility. That was the difficulty with light ground; they could cut down the count to any extent they liked; they could cut it down to nothing for fine dust by opening the iris diaphragm on the microscope.

Both methods had their drawbacks and both their advantages. Both should be used and used with discretion. Light ground was liable to err on the low side and dark ground on the high side if exceedingly fine particles were not to be considered. For practical working, dark ground was far and away the easiest. A man counting with light ground would soon get his eyes so tired that he would not be able to see anything like the number of particles which a man working with dark ground illumination would see.

Mr. H. R. Adam said Mr. Gray had stated that with the light ground they could cut down the count to anything they liked, and with the dark ground they could increase it. That was just exactly what he (the speaker) had been trying to point out. After all, it is not a matter of which is the easiest way to count, but which gives the more correct result.

Mr. R. A. H. Flugge-de Smidt said he would like to remark that some very large experiments were being carried out at the present moment on fourteen different mines, and spots had to be taken by various people on those mines and sent in to the Chamber of Mines to be counted. That meant anything from 350 to 400 spots had to be counted every day by two counters. He would suggest that that would be entirely impossible by using the separate slides of the Kotzé konimeter and was only possible by using the slides of the circular konimeter.

THE VALUE OF RAND GOLD MINING INVESTMENTS.

BY W. A. CALDECOTT & G. J. V. CLARENCE.

(Printed in *Journal*, December, 1922.)

DISCUSSION.

Mr. E. Homersham: I have read with interest Dr. Caldecott's and Mr. C. J. V. Clarence's paper entitled "Value of Rand Gold Mining Investments."

On page 101 the authors remark: "Gold mining differs from other industrial undertakings in that the product represents money and hence it is not adversely affected by any depression which may reduce the market demand and price of all other commodities, because low commodity prices tend both to reduce gold mining costs and to enhance the purchasing power of the gold produced and of the dividends distributed." All very fine and very pretty.

The authors, however, forgot to point out that the converse is also true, and that when stores, wages and other commodities rise not only are working costs increased but the actual value in the purchasing power of money distributed to shareholders automatically falls.

A comparison is then made between an imaginary investment in Consols compared with an investment in the Robinson G.M. shares at par.

Well, Mr. Chairman, it is easy to make money jobbing backwards and often figures are, as a French author whose name I have forgotten, remarks, "at the service of anybody to use as they please."

The Robinson G.M. Company was registered in 1887. The original capital was £50,000 in £1 shares, 45,000 shares being the purchase price paid to the vendors, 5,000 £1 shares being allotted for working capital.

In February, 1889, the capital was increased to £53,375 by the creation of 3,375 extra £1 shares, the fresh shares issued being used as part purchase price of six gold mining claims and one hundred acres of mynpacht acquired from the Kam Hulla Syndicate.

At this time the £1 shares were freely bought and sold at £70 to £80 each.

A few days later the company increased its capital to £2,750,000 in 550,000 shares of £5 each, the existing shareholders receiving ten £5 shares for every £1 share held by them.

The remaining 16,250 shares were sold at par to provide working capital.

It is therefore of interest to note that the actual cash working capital supplied to develop the Robinson mine was in all only £86,250. It can be readily understood that the management of this mine for many years had to use the greater portion of the profits earned for extra mine development and increase of plant in order to bring the mine to the state its prospects warranted.

It was many years before the shareholders who put up the working capital got an adequate return in dividends for the risk they had taken. These first shareholders were a favoured few and were not in a position to figure out even approximately the ultimate value of the mine, and they put up their money as a speculative mining investment, spelling their luck with "P."

By the time as a mining investment as understood by the authors the Robinson Company had found its feet, the shares were standing at anything from £7 to £10 and more. Therefore cautious investors who bought for dividends and amortisation did not have a particularly fine investment.

As I pointed out before, it is always quite easy to make money jobbing backwards, but gold mining investments on the Rand or elsewhere must always remain suited to the man who will take fair but considerable risk, and no proper comparison can be made between well secured Government Trustee stocks and mining shares, whether on the Rand or elsewhere.

What we can safely say of the Rand mines generally is that *as mining investments* they compare most favourably with similar investments on other fields; and comparatively speaking up-to-date they may be considered to wear "the white lily of a spotless life."

The authors' figures, useful as they are for valuing ordinary investments and as an assistance to the experienced mine shareholder, will I think, if used by the inexperienced layman, be a source of danger.

The author's optimistic statement on page 104 that:

"anyone conversant with the highest class of mining stock on the Rand will have no difficulty in obtaining at least 8 per cent. on the redemption fund with consequent increase in the present value of his shares" is, I consider, a dangerous statement, and special danger is in the words "anyone" and "no difficulty."

A careful review of the present position of fully developed profit earning mines of the Rand would probably show some which can be reasonably expected to improve on

present results and show in time a deserved capital appreciation, but I think the words "no difficulty" are not right or fair words to use.

The gold mining industry on the Rand and the financiers who have long been connected with it have an honourable record, but nothing is to be gained by comparing gold mining investments of whatever nature to guaranteed Government stocks.

If it were such an easy matter to place an investment value on the gold mining shares of the Witwatersrand the directors and engineers of the various groups would not, I think, carefully refrain from publicly doing so.

As a matter of fact, when in very few cases this has been publicly attempted by men with the very best available information and best intentions, it has not been particularly successful.

On page 103 the authors remark: "The prompt publication of accurate information by Rand mining companies greatly facilitates accuracy of valuation and is probably unsurpassed by the practice in this regard of any other class of industrial enterprise."

Whilst agreeing fully with the authors as to the valuable and prompt information published by all mining groups I would point out that Western Australia is far ahead of the Rand in the publication of mining information useful to the careful mining investigator.

Western Australian mines, notably the Golden Horse Shoe, publish with their annual returns assay plans and longitudinal and cross sections which are a long way ahead of anything published here locally.

In Western Australia and, I think, in the whole of Australia any shareholder has a right to inspect the up-to-date assay plans of a mine he may be interested in, and any member of the public not a shareholder, by paying a fee of one shilling has a perfect right to do the same.

I do not think therefore that we need flatter ourselves that we are a long way ahead of all other mining countries in publishing information for the public and shareholders.

Before sitting down I should like heartily to congratulate the authors on the remarks they make on page 103, reading as follows:

"The degree of accuracy with which the curves may be read is greater than either the estimated life can be calculated or the probable future dividends foretold."

This sums up, in my opinion, the whole question of the danger of the layman using

these curves to work out the investment value of the mining shares he may hold.

RECOVERY OF GOLD BY BLANKET CONCENTRATION IN SUBSTITUTION OF PLATE AMALGAMATION.

BY F. WARTENWEILER.

(Printed in Journal, February, 1923.)

DISCUSSION.

Mr. Andrew King thought the Society was to be congratulated on receiving a paper of that kind from Mr. Wartenweiler at such an opportune moment when many mines were either changing or about to change their plate amalgamation to corduroy. Mr. Wartenweiler had shown them—and they had heard from other sources also—that the total recovery, including cyaniding, was quite as good when corduroy was used as when plate amalgamation was used. Not only had Mr. Wartenweiler set at rest any fears they might have on that point, but he had also given them much valuable experimental data on various details of the process, which should considerably reduce the work on other plants changing from amalgamation to concentration, and he thought that was a very excellent feature of the paper. For example, he had given them his results on the different kinds of cloth with which he had experimented and had also given them information on the recovery obtained on the various sections of cloths. In that regard, if he remembered rightly, it was mentioned that on one or two of the bottom sections of cloth the recovery was very small; for some reason which was not given those last two sections of cloth failed to function after five hours, and he thought Mr. Wartenweiler, if he had the information, should give them a little more enlightenment on that point.

In connection with the methods described of handling the different products from the concentrator the speaker expressed the opinion that the rich concentrate consisting largely of free gold should be collected and treated separately from the middling pyritic portion and thus reduce the value of the barrel residue.

Regarding the introduction of concentration in old plants where possibly elevating capacity and plate area are limited it would possibly be of interest to members to know

that corduroy tables can be run satisfactorily with a water ratio equally as low as amalgamated plates and that the extraction (by assay) will be almost as good.

The meeting then terminated.

BOOK REVIEWS.

A MANUAL OF DETERMINATIVE MINERALOGY. By J. Volney Lewis. Messrs. Chapman & Hall (Price 16/6).—A third edition of this work has now been published and the tables revised and enlarged. A new classification of minerals based on their physical properties has been added. The blowpipe tables include about 355 minerals and the physical classification about 290; these tables are largely complementary and complete cross references facilitate their use. The arrangement and the completeness of this book are unique and render it invaluable not only to the student but also to the engineer, geologist and chemist. Identification of any mineral may readily be made, after carrying out a few simple tests, by reference to the tables. Minerals are classified under a number of distinct physical and chemical properties such as crystalline form, hardness, streak, cleavage and simple wet and dry chemical reactions, thus reducing the work of identification to a minimum. The book is a valuable companion to works on descriptive mineralogy and fills a long-felt want; it will inevitably find its way to that part of the technical bookcase where the best-thumbed volumes are.—(R.A.C.)

FUEL IN SCIENCE AND PRACTICE. Edited by R. V. Wheeler, D.Sc., Professor of Fuel Technology, University of Sheffield, and J. V. Elsdon, D.Sc.; published by the Colliery Guardian Co., Ltd., 30 and 31, Farnival Street, Holborn, E.C. 4 (Price 10/6 yearly). Vol. II., No. 1.—This journal has commenced its separate and independent existence as a journal with the publication of the first number of the second volume, its first volume having been completed as a supplement to the *Colliery Guardian*. Its function is stated to be a journal of the scientific and economic use of fuels, being the record of the Coal Research Club. A journal devoted to this object cannot possibly fail to enlist the sympathy and active support of all those who are interested, however remotely, in the conservation of the world's chief fuel supply, namely, coal, the use of which has hitherto been marked in most countries with an extravagance born of ignorance of the limited supplies at disposal and the numerous valuable products which may be obtained from it. The cries and pleadings of scientific men to stop the waste, which have been uttered during this century, are apparently at this late hour beginning to penetrate the obscure vision of those responsible for the nation's welfare as marked by the establishment of the Fuel Research Laboratory, under the direction of Sir George Beilby. The present journal is the complement of that step in that it will bring the work of fuel study and research within the ken of all interested in this vital economic problem. Owing to the fact that the

Coal Research Club have definitely adopted it as its mouthpiece it must necessarily possess an authority of unrivalled standing. One of its editors, Prof. Wheeler, has a world-wide reputation for his researches on coal, and this fact alone affords sufficient evidence that its articles and contributions will be of the first rank. This is borne out by a glance at the contents of the present number. That such a journal deserves the heartiest welcome and support is a mere truism, and we trust that it will meet with every possible success in both its career as a scientific journal and its noble mission to the nation.—(J.A.W.)

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

DETERMINATION OF FERTILISING VALUES.—In a paper on "The Valuation of Insoluble Phosphate by Means of a Modified Citric Acid Test," a plea was put forward for a modification of the present generally accepted test—referred to as the official test—for the valuation of insoluble phosphate. It was suggested that the present official citric acid test exaggerates the inferiority of the lower citric soluble slags, and that the results from pot experiments, upon which the five-gramme test of Wagner was based, are not necessarily paralleled by the results obtained from field experiments. Such experiments had shown that typical fertilisers had a higher fertilising value than was indicated by the official citric acid test, viz., that in which five grammes of phosphate, ten grammes of citric acid, and 500 cc. of liquid are taken and shaken for thirty minutes, and the proposal of the authors, based on many experiments carried out at Essex experimental centres, was that one gramme of phosphate instead of five grammes should be taken. In support of this, figures were given which indicated that the one-gramme solubility test gave results which more closely followed the field experiments than do the results from the five-gramme test. It was admitted by the authors that this modified method had no scientific basis, but was a rough-and-ready method of getting at the true value of the various phosphates on the market.—G. S. ROBERTSON and F. DICKINSON, *Iron and Coal Trades Review*, December 8, 1922, page 856. (J.A.W.)

BROWN COALS AND LIGNITES.—In his Cantor lectures on this subject, before the Royal Society of Arts, Professor W. A. Bone, F.R.S., dealt with the questions of low-temperature carbonisation of these coals and their use for steam-raising purposes. He said he did not advocate the low-temperature carbonisation of brown coal and lignites on the commercial side, and he did not know of any large-scale plant at work at present. The objects to be aimed at in subjecting these fuels to preliminary treatment by carbonisation were: (1) to up-grade and stabilise their character as fuels; (2) to separate from the volatile decomposition products any condensable oils, and also ammonia,

if ammonia be recovered in sufficient quantities to pay for its recovery; and (3) to utilise any surplus gas available. Of these objects, one of the most important in the immediate future would be the first. Apparatus had been constructed, and was in use at South Kensington, for testing these fuels. Working with this plant, certain results had been obtained with a dry, brown coal containing 30 per cent. oxygen, when carbonising at 375, 500, 700 and 850 deg. Cent. There was no evolution of any combustible gas up to 375 deg., except a very small quantity of carbon monoxide. At 500 deg., whilst there was also the evolution of carbon dioxide and water vapour, methane began to appear in the products, and at this temperature about two-thirds of the available oxygen was obtained in the form of oxides of carbon, and steam. Passing up to 700 deg. temperature, the evolution of carbon dioxide fell off, but the evolution of methane had gone on rapidly increasing; hydrogen also appeared, and at this temperature five-sixths of the oxygen came off as oxides of carbon and steam. Going to a temperature of 850 deg., there was a very large production of hydrogen, and figures obtained from a recent brown coal were very instructive as to the manner in which these coals carbonised at low temperatures. A great deal of nonsense was talked as to brown coal and lignites being capable of producing very large amounts of valuable oils and tars having unheard-of properties and qualities. He believed that the low-temperature carbonisation of these fuels would be largely from the point of view of improving the residual fuel. One fact brought out was that by going to a temperature of 850 deg. the whole of the oxygen was got rid of and a very good type of charcoal fuel was obtained; he believed it would be from that point of view that the carbonisation of these fuels would be taken up. The oils, for the most part, were of the naphthene or hydro-benzene series, and there was very little of the paraffin hydrocarbons. When the higher fractions were cooled, a certain amount of waxy material was obtained, which was hydrocarbon in character.

Oils.	Density.	Lb. per ton of coal.
Light spirits	0.85 ...	13.8
Light oils	0.88 ...	19.8
Intermediate oils	0.95 ...	6.7
Heavy oils	1.0 ...	22.5
Pitch	1.1 ...	23.7

If the solid residue from the carbonisation process were mixed with about 5 per cent. of pitch, it made a valuable smokeless fuel, there being about 15 per cent. of volatile matter remaining; if the temperature of carbonisation was kept within the limits mentioned, viz., up to 550 deg. Cent., and if briquetted into walnut size, it formed an excellent fuel for use with chain-grate water-tube boilers. The difficulty was that there was not sufficient pitch obtained from the process to provide the percentage required for briquetting. The shortage of pitch and asphaltic residues which had the same characteristics as pitch was a very great hindrance to the development in this direction. Another promising field of use for these coals was in a pulverised form for use in boilers. In using these fuels in this form, however, care

must be taken to leave a certain amount of volatile matter in the residue after carbonisation, and apparently from what one read of the work being done in America, 10 to 12 per cent. volatile matter was the proper amount. Again, this residue formed a good fuel for gas-producer purposes, but even then it was advisable to briquette it before use. His experience was that there was no difficulty in gasifying either raw lignite or the carbonised residue in gas producers, especially of the ammonia-recovery type. In dealing with prospects for utilising brown coals, Professor Bone said the great drawback hitherto had been that engineers had approached the problem from the point of view of utilising a new fuel in boilers of the ordinary construction, instead of re-designing the boiler and furnace for the special purpose for which it was to be used. He had formed the conclusion, after making several boiler tests, that a satisfactory rate of combustion for a fuel of this kind was 40lbs. per hour per sq. ft. of grate area, compared with a maximum of 30lbs. with bituminous coal. Therefore, they must have a combustion chamber big enough to ensure vigorous combustion of the very large amount of volatile hydrocarbons given off from the coal.—W. A. BONE, *Colliery Guardian*, December 29, 1922, page 1587. (J.A.W.)

INADEQUACY OF "A.R." TEST FOR ALKALIS IN CALCIUM CARBONATE.—The authors, whilst investigating the composition of a series of glasses consistently obtained totals varying from 107 to 108%. The purity of the reagents used was suspected, though these were guaranteed to be "A.R." quality—the standard of purity regarded necessary for analytical work.

The "A.R." standard of examination of calcium carbonate depends on the water solubility of the contained alkalis, and calls for an amount not exceeding one milligram of residue to be obtained when five grams of the substance are boiled in 50 c.c. of water, filtered, and 25 c.c. of filtrate, evaporated and ignited.

Experiments indicated the presence of larger quantities of alkali carbonates, of this only 9 to 15 milligrams were dissolved in the first boiling with 50 c.c. of water; in all five extractions were given, and the alkaline carbonates extracted amounted to 17 to 29 milligrams from five grams of calcium carbonate. As this comparative large amount failed to account for the extraordinary totals, the Lawrence Smith method which was used for the determination of the alkalies present in the glass was applied to the calcium carbonate. Five grams, mixed with half a gram of ammonium chloride were heated in a covered platinum crucible to a dull red heat for one hour. The aqueous extract after complete removal of the lime was acidified with hydrochloric acid, evaporated, and ignited.

The chlorides obtained were almost pure sodium chloride containing only a trace of potassium chloride, and were equivalent to 112 to 118 milligrams of sodium carbonate.

Since only about 10% cent of the contained alkali carbonates are removed by one extraction with water, the "A.R." method is obviously inadequate and requires revision.—WILLIAM SINGLETON and HOLWELL WILLIAMS, *The Analyst*, June, 1922, Vol. XLVII, No. 555, p. 252. (H.D.B.)

METALLURGY.

NEW USES FOR NON-FERROUS METALS.—Chromium Plating. The present day interest in stainless metals should attract attention to the possibilities of chromium deposits. Bunsen in 1854 obtained continuous deposits of this metal in the laboratory by electrolyzing chromous-chromic chloride solution with high current density.

A company has recently been formed in Berlin, Elektro-Chrom G.m.b.H., to exploit the process worked out by Dr. Erik Liebreich. From the published descriptions it is difficult to see any real novelty, but the inventor claims to have succeeded in obtaining strongly adherent deposits on iron, copper, brass, etc. Thin deposits have a bright surface but matt grey powdery layers of coherent form and offering complete protection against the attack of the basis metal can be obtained of any desired thickness and with 50 per cent. of the theoretical efficiency of the electric current. The author's chief contribution appears to lie in his investigation of the periodic fluctuations of the current when chromic acid and chromate solutions are electrolysed, and his explanation of this as due to four different stages of reduction of the chromium salts. Finally in a chromous salt solution a steady deposition is obtainable at current densities of .02 to .10 amps per square centimetre. No mention is made of the earlier and very thorough investigations of B. Neumann and G. Glaser (*Zeitschrift für Elektrochemie*, 1901, Vol. 7, p. 656), who obtained from chromic chloride and chromic sulphate solutions deposits of chromium with 50 to 85 per cent. current efficiency at .10 to .20 amps. per sq. cm.

This older work, so far as one can see, deprives the invention of much of its novelty. Any application of the deposition of chromium would doubtless lead to important commercial results and find wide applications on account of the resistant qualities of chromium to oxidation.

Cadmium for Rust-proofing. The Udylite Process Company, Kokomo, Ind., U.S.A., has patented a process of electro-deposition of cadmium and subsequent heating for several hours at 150–200 deg. C. It is claimed that a deposit one-half to one-fifth of the thickness of zinc is sufficient, and that the cost of electric current is lower. The process had been applied to rust-proofing locks, hardware, etc., and is said to be in use in 27 plants in the U.S.A. A pamphlet just issued by the Udylite Process Co. gives the results of salt-spray corrosion tests in support of the claims made. (See *Monthly Review*, American Electroplaters' Society, June and July, 1922; also U.S.A. Geological Survey: "Cadmium in 1921.")—Bulletin of the British Non-Ferrous Metals Research Association, October, 1922, p. 16. (F.W.)

BOOK REVIEW. — CYANIDING GOLD AND SILVER ORES. By H. Forbes Julian and Edgar Smart; revised by A. W. Allen. London: Charles Griffin and Co., Ltd., Exeter Street, Strand, W.C.2. (Price, 36s. net.)—This is the third edition of "Julian and Smart." The first-named author went

down in the "Titanic," and Smart was murdered by natives in Africa. For some time before his death he was associated with the reviser in preparing the present new edition of this important treatise, which will be found to contain much new material relating to the process, both as regards theory and in operation.—*Iron and Coal Trades Review*, September 29, 1922, p. 468. (J.A.W.)

PRESENT STATUS OF THE ELECTRIC FURNACE IN REFINING IRON AND STEEL.*—There are now nearly 1,000 electric furnaces in America and Europe, not quite half of them in the United States and Canada. Constant interest in these developments from the beginning leads one to feel that the pioneer writers were not over-enthusiastic, and that their claims and predictions have, in general, been fulfilled. It will surprise many to know that Italy has about 180 electric furnaces for steel melting, that 27 of them are from 15 to 25 tons capacity, and in 1921 her tonnage of electric steel was second only to that of the United States, and reached a new high mark for that country. The annual productive capacity there is about 1,000,000 tons, and Dr. Giolitti recently told the author that some of these furnaces were operating at unusually high speed and with great economy of electrode consumption, as low as 6½ lbs. per ton for cold melting. For installed capacity, Italy ranks ahead of Germany, England and France, and second only to the United States.

The rapidity with which electric furnaces have been installed within the last decade all over the world calls for some analysis as to cause. After ten or twelve years of invention and pioneering there were about 125 furnaces in the world in 1912. At this time, Germany led with nearly one-third of the total number. Today, as nearly as may be estimated, there are 1,000 furnaces, 388 of them being in the United States. Accurate statistics have been extremely difficult to secure during late years, but according to Dr. Richard Amberg there are 65 furnaces in Germany engaged in the manufacture of ingots, with a yearly productive capacity of 430,000 tons and an unknown number of furnaces making steel castings with a capacity estimated at 300,000 tons per annum. The total number of electric furnaces in Germany may be estimated at about 100 to 110.

The reasons for the world-wide expansion of electric steel-making are three:—

(1) Cheapening of wholesale power rates, due to hydro-electric and improved steam-plant developments. Thus it is now commercial to use electricity for melting, whereas the original promoters of arc furnaces felt that their use would be of necessity confined to refining of metal pre-melted by the older processes.

(2) The extreme flexibility and adaptability of electric furnaces to a wide range of uses. It has been shown by experience that they may be successfully used for melting cold charges

*From a Paper read before the October Meeting of the American Iron and Steel Institute. The Author is President of the Crucible Steel Company of America.

or refining liquid charges, for making ingots or castings, and for melting ferro-alloys. They may be used alone, or in conjunction with the Bessemer or open-hearth, or both. They may be operated acid or basic. They may be used in conjunction with the blast furnace or cupola for making grey-iron, malleable and semi-steel castings. For foundry use particularly, the small units are advantageous for making frequent small heats of steel or iron castings.

The most popular size of electric furnace in the States is of 6 tons capacity, but furnaces from $\frac{1}{2}$ to 40 tons capacity have proved equally successful. In furnaces of 6 tons, or a little larger, hand charging is general, but in the larger sizes either mechanical charging of cold materials or the use of hot metal charges is usual. Duplexing of open-hearth steel is practised in many of the larger units, while triplexing is done at the great installation of the Illinois Steel Company.

All of the manifold methods of operation are possible with the use of arc furnaces, which are by far the most frequently used in the States and abroad. Of the different types of arc furnaces in the United States nearly one-half are of the Héroult type and considerably more than one-half of the productive capacity is represented by them.

The electric furnace has small possibilities in the States for the manufacture of pig-iron from the ore, but during the war period several furnaces were used to make so-called synthetic pig-iron from turnings and borings and other light scrap, in the United States, Canada and France. In Sweden, Norway and Italy, where metallurgical fuel is very dear and electricity is cheap, electric smelting of ores is an established industry. The world's production for 1921 is placed at 377,900 tons, and it will be recalled that the year 1921 was not a good year for high records.

(3) Quality of products. A new process to succeed must be cheaper in operation or produce a better quality. The cost of electric steel is rarely lower than open-hearth and never lower than Bessemer, and therefore its success is presumably due to its producing a generally superior product. Of course there are especially favoured localities of peculiar market conditions which warrant the installation of small electric furnaces where open hearth and Bessemer installations would be out of the question. In the same localities and markets electric furnaces do operate successfully alongside of large tonnage plants and under such conditions quality must be the principal reason for success rather than low cost.

The electric furnace is a potential source of clean steel, which is more highly appreciated than formerly, and the electric product is opportune to meet the new and exacting requirements for ordnance, automobiles and aeroplanes, and other devices in which alternating stresses are very severe. The results of electric furnace ordnance steels made at Charleston, W. Va., in the largest furnaces in the United States were due to clean steel and freedom from oxides, sulphur and phosphorus. These results show that large units, with proper handling, can produce very high-quality steel, and we see

no reason, now that larger electrodes can be made of dependable quality, while a 60- or 80-ton furnace cannot be expected to give relatively as satisfactory results. The electric steel rail is still a desired possibility. The increased demands made upon materials of construction call for new methods for meeting those demands.

Of mechanical and electrical refinements there have been many, all in the nature of improvements in regulation and economy. Among these may be mentioned two of American origin, J. A. Seede's automatic electrode regulator, and E. T. Moore's peak-load regulator. The principle of dual, or rather multiple, voltages was embodied in our original installation in 1906, but its metallurgical significance was not so apparent as it was later when we installed a process employing a 220-volt arc and with considerable difficulty succeeded in persuading the inventor that provision for a lower voltage for use during the refining period must be provided. The desirability of relatively high voltage for melting and low voltage for refining is now generally recognised. Many improvements have been made as the shortcomings of the earlier furnaces appeared, such as well-fitting doors, water-cooled arches, better electrodes and holders and economisers to cut down oxidation and waste of the electrode.

The electric furnace is also a recognised factor in melting non-ferrous alloys, such as brass and Monel metal, as well as special alloys such as "nichrome," "rezistal," "stellite," stainless steel, manganese steel and high-speed steels, besides an endless variety of the simpler alloy and carbon steels from the mildest to the hardest tempers. When users acquire a full appreciation of what clean sound steel means in terms of national efficiency, safety and economy we shall see more rapid growth than has as yet been seen. Its usefulness to engineering and industry has just begun.—JOHN A. MATTHEWS, *I. and C.T.R.*, November 17, 1922, p. 716. (J.A.W.)

CHANGES OF ADDRESS.

CHILTON, J. *l/o* Modderfontein East, Ltd.: P.O. Box 186, Johannesburg.

GRAHAM, W., HASTIE, *l/o* Abbontiakoon Mines, Ltd., Tarquah, West Africa: c/o R. Urquhart, Esq., P.O. Box 108, Salisbury, Rhodesia.

KRAUSE, H. L., *l/o* New Primrose G.M. Co., Ltd.: Witwatersrand G.M. Co., Ltd., P.O. Box 1, Knights.

MORRISBY, P. T., *l/o* Witwatersrand G.M. Co., Ltd.: Clunie, Lower Park Drive, Parkview, Johannesburg.

TURVEY, W. E., *l/o* Robinson Deep, Ltd.: P.O. Box 67, Johannesburg.

WALLER, F. H. G., *l/o* Kenilworth: Robinson Deep, Ltd., P.O. Box 1392, Johannesburg.

WILKES, H. R. S., *l/o* c/o United Party Club, Johannesburg: The Assay Office, Nourse Mines, Ltd., P.O. Box 32, Denver.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA

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Vol. XXIII.

APRIL, 1923.

No. 10.

Proceedings
AT
Ordinary General Meeting,
21st April, 1923.

The Ordinary General Meeting of the Society was held in the Assembly Hall, Scientific and Technical Club, Johannesburg, on Saturday, the 21st April, 1923, at 8 p.m., Prof. G. A. Watermeyer (President) in the chair. There were also present:—

22 Members: Messrs. C. J. Gray, R. A. Cooper, H. Pirow, J. M. Thorburn, John Watson, F. Wartenweiler, A. Whitby, H. A. White (Members of Council), J. A. Boyd, W. S. Gordon, G. B. Hopwood, G. R. Munro, W. W. Olver, T. K. Prentice, C. B. Simpson, R. A. H. Flugge-de Smidt, S. de Smidt, J. J. R. Smythe, J. Thorlund, A. J. Walton, H. R. S. Wilkes, and A. A. Wood.

4 Associates: C. E. Deakin, J. Gibson, H. G. Kay and G. C. Wilson.

5 Visitors, including Dr. A. Mavrogordato, and H. A. G. Jeffreys (Secretary).

MINUTES.

The Minutes of the Special General Meeting held on the 10th February and those of the Ordinary General Meetings held on the 17th February and 17th March, 1923, were confirmed.

GENERAL BUSINESS.

ASSOCIATED SCIENTIFIC AND TECHNICAL
SOCIETIES OF S.A.

The President called the attention of members to the draft Rules of the Associated Scientific and Technical Societies of South Africa Provident Fund, which the

Associated Societies proposed to establish. The object of the Fund was to provide, on the death of a member, a sum not exceeding £100, to be paid to the nominee or nominees of such deceased member. The actual sum payable would be regulated by the amount received through levies, less administration expenses. Members in good standing with their Constituent Societies, and under forty-five years of age, would be eligible for membership of the Fund, and members above that age, up to sixty years, would also be admitted, the maximum benefits being arranged on a sliding scale according to age. Applicants for membership would be required to pay an entrance fee of five shillings, and upon the death of a member a levy would be made of one shilling per member. To facilitate the collection of levies, members on joining would pay five shillings in advance to cover future levies, and further like sums would be payable as the funds become exhausted.

He would like to have an expression of opinion from the members present and to know if the scheme met with their approval.

Mr. C. J. Gray suggested that time should be given for members to consider the proposed scheme and that the matter be brought up at the next meeting.

This was agreed to.

DANCE COMMITTEE REPRESENTATIVES.

Messrs. A. A. Wood and J. A. Boyd were appointed to act as the Society's representatives on the Associated Societies' Dance Committee.

THE CLUB'S LICENCE.

The President drew the attention of members to the Rule of the Scientific and Technical Club which required that on the introduction of visitors their names must be

entered in the Club Visitors' Book. It was essential that this rule be observed as it was one of the conditions of the Club licence.

"A TEXT BOOK OF RAND ASSAY PRACTICE."

The President announced that one of their kindred Societies, the South African Association of Assayers, had issued a Text Book of Rand Assay Practice, price 15s. He would like to draw the attention of members to its value and to the fact that the edition is limited to three hundred copies.

NEW MEMBERS.

A ballot was taken for the election of new members, and the following gentlemen were declared unanimously elected:—

ANDREWS, F. B., P.O. Box 5627, Johannesburg: Solicitor.

BELL, JOHN, 17, Isipingo Street, Bellevue, Johannesburg: Merchant.

DAUDE, E. F. B., Owyhee Hotel, Boise, Idaho, U.S.A.: Mining Operator. (Transfer from Associate Roll as from 1st July, 1923.)

HOPWOOD, G. BROWN, P.O. Box 103, Crown Mines, Johannesburg: Dust Inspector.

KRAUSE, H. L., Witwatersrand G.M.Co., Ltd., Knights: Mine Manager. (Transfer from Associate Roll as from 1st July, 1923.)

WHITEHOUSE, JAMES, P.O. Box 1169, Johannesburg: Consulting Engineer. (Transfer from Associate Roll as from 1st July, 1923.)

The Secretary announced that the following gentlemen had been admitted by the Council as Associates:—

CARRUTHERS, JAMES THOMAS, 115, Railway Street, Mayfair, Johannesburg: Reduction Works Smelter.

KAY, HERBERT GEORGE, P.O. Box 1421, Johannesburg: Mine Manager and Surveyor.

and as a Student:

CLUCAS, W. H., Witwatersrand University, Johannesburg.

FURTHER NOTE ON RETURN TO CAPITAL INVESTED IN THE WITWATERSRAND.

By R. A. LEHFELDT.

In my paper "The Return to Capital Invested in the Witwatersrand"* an attempt was made to work out the precise amount of the profits made by mining gold on the Witwatersrand.

The period chosen for the investigation was from the 30th June, 1907 (a time when Stock Exchange quotations were fairly steady) to the latest available date—31st December, 1921.

The method adopted was to suppose an investor, who, on 30th June, 1907, bought all the shares in the Witwatersrand mines at the market price, who subsequently supplied all the new capital that was supplied, received all the dividends and repayments, and sold out his interest at market price on 31st December, 1921. What return would the imaginary investor have got on his money?

The average yield is such a rate of interest that all the outgoings, accumulated at that rate, to a given date will balance the income similarly accumulated. A rate was therefore chosen arbitrarily, and all the sums, in and out, were multiplied by factors that would represent accumulated interest to the

date chosen, 31st December, 1921. Six per cent. was actually tried, and it was then found that the income appeared to exceed the outgoing. On recalculating with seven per cent. interest the position was reversed, so the true rate lies between the two, and was found by proportion to be 6.20 per cent.

That is the annual percentage return to the capital invested in the Witwatersrand gold mines treated as a whole.

It would take too long to work out a similar annual percentage return for each individual mine; a simpler method of presenting the essential facts was therefore adopted in this case.

To make the method clear, let us again avail ourselves of the concept of the imaginary investor—in this case not an investor who bought all the Witwatersrand shares, but one who bought all the shares of an individual mine at market price, who subsequently supplied all the new capital that was supplied, received all the dividends and repayments, and sold out his interest at market price on December 31st, 1921. What would have been the ratio of his incomings to his outgoings over the whole period?

* This *Journal*, January, 1923, page 129.

To obtain this ratio an arbitrary rate of interest was chosen (6 per cent. per annum) and all the incomings on the one hand, and all the outgoings on the other, were accumulated at that rate to a given date (December 31st, 1921). All that was now necessary was to work out the ratio between the two.

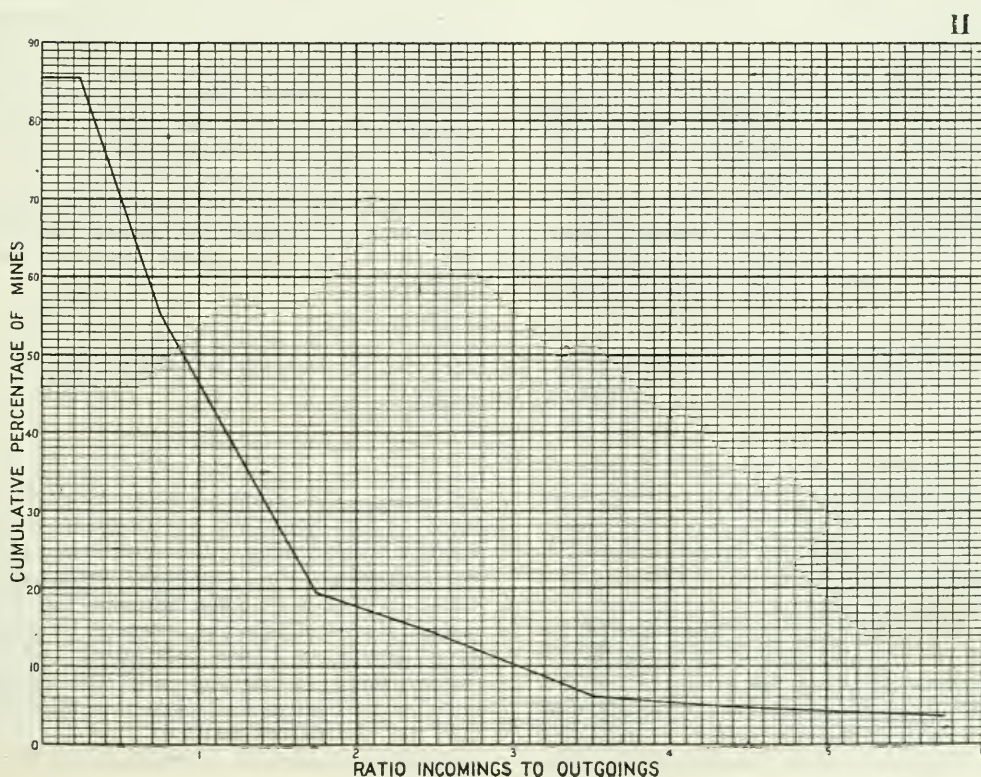
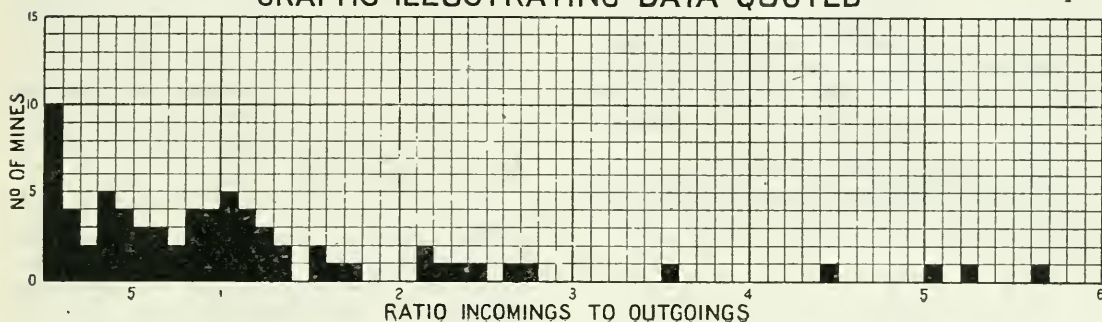
Let us take the Brakpan Mines as an actual example:—

Name of Mine.	Outgoings Accumulated.	Incomings Accumulated.
Brakpan Mines	£2,228,232	£5,971,995

Ratio of incomings to outgoings:—

$$\frac{5,971,995}{2,228,232} = 2.68$$

GRAPHS ILLUSTRATING DATA QUOTED



That is, the Brakpan Mines have yielded a total return 2.68 times as great as the total amount invested in the mine would have amounted to at 6 per cent.

The Geldenhuis Deep provides us with another example.

Name of Mine.	Outgoings Accumulated.	Incomings Accumulated.
Geldenhuis Deep	£6,626,066	£3,426,706
Ratio of incomings to outgoings:—		
	3,426,706	
	= .52	
	6,626,066	

That is, the Geldenhuis Deep has yielded a return, the total amount of which is only .52 of what the investment in the mine would have amounted to at the same rate as before.

The following table shows the ratios of incomings to outgoings in the different mines and the number of mines in each group:—

Ratio Incomings to Outgoings.	Number of Mines in each group.
5.01 to 5.75	3
4.01 to 5.00	1
3.01 to 4.00	1
2.01 to 3.00	7 (i.e., only 12 mines have yielded more than double the investment.)
1.51 to 2.00	4
1.01 to 1.50	14 (i.e., altogether 30 mines have returned more than the investment.)

Ratio Incomings to Outgoings.	Number of Mines in each group.	
0.51 to 1.00	16	(i.e., of the rest, 16 have yielded over half the investment.)
0.01 to 0.50	25	(i.e., 25 have returned only half or under half the investment.)
Nil	12	(i.e., 12 mines have yielded no return at all.)
No. of mines	83	

The results obtained are illustrated graphically in the accompanying diagrams, which show the ratio between incomings and outgoings, as explained above.

The President was sure members would accord a very hearty vote of thanks to Dr. Lehfeldt for the trouble he had taken in presenting the note. It was very interesting and, to his mind, gave them more information than did the authors' previous paper, where he simply worked out the average percentage taken over all the mines, which he found to be 6.2 per cent. The note showed that a number of mines had given quite a large return. They would see that the extreme was in the neighbourhood of 5.75 times the capital invested, which was very satisfactory.

SOME METHODS AND RESULTS IN DUST-SAMPLING ON THE WITWATERSRAND.

By A. MAVROGORDATO.

(The following is an abstract of the communication made to the Society at the meeting. The paper will shortly be published in full by the South African Institute for Medical Research.)

This enquiry, which has been carried out by the group whose names are given below, appeared to be called for owing to the lack of general confidence in the methods used for dust-determination and a consequent state of doubt as to conditions actually

obtaining. The unrest was particularly related to the fact that gravimetric returns, though reliable within their limitations, were not affording the necessary information, while lack of agreement between the konimeter returns of different observers or groups of observers, and not infrequent lack of the expected correlation with simultaneous gravimetric returns had shaken confidence in this the alternative method.

It is only dust of a certain nature and

within certain limits of size that is of importance, and such dust has been called *phthisis-producing dust*. For our purposes phthisis-producing dust may be defined as dust of free silica that is not above 5 microns or below half a micron in diameter. An ideal method should estimate the content of the air in phthisis-producing dust, but no single method affords such information.

Gravimetric methods, such as the sugar-tube, used with the Moir and Johnston technique, are very efficient, and their return is confined to insoluble mineral matter of above a quarter of a micron in diameter, but they fail because they do not afford essential information as to the size of particles, and, without large size of sample, are useless with low concentrations.

Konimetric methods fail because, depending as they do upon optical examination, they afford no information as to the identity of small particles, and the air underground is often rich in water-soluble and other particles that are neither silica nor insoluble mineral matter.

In the examination of the konimeter sample, putting aside fallacies due to the presence of particles other than silica, the light-ground or method of examination by direct light, as used, may err on the low side by missing important small particles, while the dark-ground or method of examination by oblique light may err on the high side by introducing unimportant minute or even ultra-microscopic particles.

Lack of correlation between konimetric and gravimetric returns can be accounted for as follows:—

(a) The lack of correspondence is more apparent than real. The accepted standard is that put forward by the Miners' Phthisis Prevention Committee (100 particles per c.c. to 1 mg. per c. metre), and was based on light-ground counting. As was pointed out at the time the number of particles will vary with their mean diameter, and this will vary with type of work. It is probable that the mean diameter of particles is now less than when these determinations were made.

(b) The konimeter gives total particle content whereas the sugar-tube gives insoluble mineral content. Variations in concentration of water soluble particles are independent of variations in insoluble mineral particles.

(c) When dark-ground illumination is

used the konimeter discloses particles not caught by the sugar-tube. As to whether such particles are there or not is a matter of fact: as to whether such particles are phthisis-producing or not is a matter of opinion.

All sampling methods are subject to errors and variations due to the method and errors and variations due to the observer. It is frequently stated that, as dust is not produced at a uniform rate, and when produced is not distributed in a uniform manner, a konimeter "grab-sample" can not possibly afford the necessary information as to the amount of dust to which a workman is exposed in the course of a day. We had to satisfy ourselves—(a) that the konimeter did afford this information. (b) that it gave or could give reasonably uniform returns in the hands of different observers.

With regard to methods we have satisfied ourselves:—

(a) That if attention be given to uniformity of jet, which depends upon uniformity of orifice and strength of spring, then two or more simultaneous samples taken on different konimeters correspond reasonably well. There is as good correspondence between a Kotzé and a circular konimeter as between two Kotzés or two circulars.

(b) Sir Robert Kotzé proved that, of the dust in the air that entered it, a konimeter caught over 85 per cent. up to 500 particles per cc. These results were confirmed by the Dust Sampling Laboratory of the Chamber of Mines and by Dr. Owens of the British Home Office.

(c) The question as to whether the dust in the air that enters a konimeter corresponds to or varies with the dust in the air at large can be tested against gravimetric samples of considerable volume taken over a considerable space of time. The expected correspondence was worked out for the Miners' Phthisis Prevention Committee, and some 6,000 simultaneous samples taken by the Chamber of Mines show that, on the whole, this correspondence is found in practice. A further test can be made by counting the particles in a sugar-tube sample and comparing them with the counts on konimeter samples taken at the same time. Such tests have been made in America, Australia, and here. The konimeter count is about 18 per cent. of the sugar-tube count, and, on the whole, varies uniformly with it.

One may say then that, as far as the method goes, the konimeter does afford the necessary information. It gives a measure, but not the absolute figure of particle concentration.

Variations due to the observer are beyond all comparison more serious than variations due to the method; they are not related to the konimeter but to the use of the microscope. There is, at present, no uniformity in methods of examining the spot.

Some observers use dark-ground and some light-ground. Correspondence is impossible.

No particular attention is given to source of light with light-ground counting. This makes for differences.

While care is taken to use similar objectives and eye-pieces no consideration is given to condensers. This is of the first importance, particularly with dark-ground counting.

When a sector-ruled eye-piece is used differences in centring the spot may affect the count.

Different observers have different standards of what to count and what to leave out. Some count every particle, some only count what they "know" to be silica, some only leave out what they "know" not to be silica.

All these sources of variation can be eliminated by a standard method of microscoping the spot. In addition we must accept the fact that acuity of vision varies with different people, and that fatigue, both mental and retinal, affects counts.

With uniform methods the konimeter counts vary uniformly with the same observer, and should vary sufficiently uniformly for routine work and form a reliable measure of the total dust-content of the air, though not of its content in phthisis-producing dust.

Conditions that are safe by the konimeter *are safe*, conditions that are dangerous by the konimeter should be assumed to be dangerous until proved to be safe by controlling studies.

What is required of the konimeter is that it should put us on the track of dangerous conditions and operations, and it is suggested that Sir Robert Kotze's method of examining the spot in situ on the konimeter will give all the information that the konimeter per se is capable of affording. This method should be given a thorough trial before we commit ourselves to the standardi-

sation of elaborate and expensive instruments and methods of illumination.

The katathermometer gives valuable information as to general conditions and often as to dust conditions.

With a high temperature and good cooling-rate due to through ventilation, dust returns are good, and with a linear velocity of 60ft. per minute dust deliberately raised is quickly removed.

If a good cooling-rate is secured by "puddling" the air, dust concentration is, usually, not reduced, and may be increased.

If cooling-rate is low, dust conditions are apt to be unsatisfactory because dust raised in a few minutes tends to persist throughout the shift owing to lack of air movement.

In addition to mechanical tests, valuable information can be secured by keeping small animals underground exposed to various conditions.

(a) One can compare their duration of life underground with the duration of life of control animals above ground.

(b) One can compare their duration of life under different conditions underground.

(c) One can learn whether they are developing silicosis.

If we are still producing miners' phthisis, it is not because the mean dust-content of the air is round about 250 particles per c.c., or 2 mg. per c.m., but because workers are occasionally exposed to really heavy concentrations.

Since the majority of cases of miners' phthisis occur among machine-users, and since machine-users are in the minority among men employed underground, one would expect some relation between "dust-floods" and improper use of machines. A common cause is "tube trouble."

Since machine-boys spend much more of the shift behind the drill than does the white miner, other things being equal, if drilling is still producing miners' phthisis the machine-boy should develop signs of silicosis before the white miner and serve as a danger signal. Something might be learnt if it were possible to secure a group of machine-boys with reliable records as to the duration and nature of their occupation.

Under some circumstances one should be able to drill with exhaust hoods on the drills, and such an expedient is worth consideration when through ventilation cannot be secured.

There is no difficulty in dealing with over 90 per cent. of the dust; this 90 per cent. includes all visible dust, but not all dangerous dust. Water alone cannot be relied upon to produce safe conditions owing to the difficulty in wetting really small particles. Colloid sprays are more efficient than water sprays, and, while colloid spraying and washing down is not of general application, there may be scope for its use.

The President was confident members would join with him in giving Dr. Mavrogordato a very hearty vote of thanks for his exceedingly interesting lecture. He thought they would agree with him that the manner in which the lecturer had handled the problem, the way in which he had ferreted out the main points of inconsistency, the way that he had brought his mind to work on the carrying out of all the tests and the results which he had attained, must receive their utmost admiration. He would ask them to give Dr. Mavrogordato a hearty vote of thanks.

Dr. Mavrogordato thanked members very much for the kind way in which they had received the communication. He must again insist that it was by no manner of means his own work. He was merely a mouth-piece. There had been a crowd of them working together. Taking the whole work, there had been the Mines Department—Mr. Ray; the Mine Officials—Mr. Innes and Mr. Pulford; the Chamber of Mines—Mr. Boyd and Mr. McEwan; and, outside the Mining Industry directly, there had been Dr. Moir of the Government Laboratories, Prof. Cluver of the University, and himself from the Institute. It had really been a combined investigation which he had had the privilege to communicate to them that night. He again thanked members for their kindness.

Mr. C. J. Gray said he had listened with great interest to the very clear exposition which Dr. Mavrogordato had given them. His own views on the more important points were quite in accord with those that had been expressed that evening. The differences between the lecturer and himself were on matters of minor detail.

There was one question which might be of a certain amount of interest to some of them. Dr. Mavrogordato had said that the sugar tube was a method of filtration, while the konimeter was a method of precipitation.

A short time ago Sir Robert Kotzé had pointed out to him that there was really no difference in method. The grains of sugar in the sugar tube were so large in comparison with the dust particles that they could not act as a filter. The air passing through the interstices between the grains of sugar would impinge on other grains, which must be slightly moist, or they would not catch the dust; it was simply a series of impacts of the particles in the air passing through the sugar tube which gave the result.

Then Dr. Mavrogordato had referred to the fact that the konimeter was not so widely known as the sugar tube. It was more widely known in America and Australia, for instance, than might be gathered from the reference. He knew from what Dr. Mavrogordato had said that he was aware of the considerable amount of work that had been done with the konimeter in Australia. He (the speaker) saw an important paper that day by Messrs. P. H. Warren and T. A. Read, of Broken Hill, in which they had compared the results obtained in dust determination by different methods. They came to the conclusion that the konimeter caught only 17.4 per cent. of the dust in comparison with another method, the Read water-spray sampler which they had devised. Presumably the instrument used was not the original Kotze, and light ground illumination was used in counting, but he (the speaker) thought that it was generally realised now that they did not catch all the dust with the konimeter; all they could say was that the dust that they caught and counted was present, there might be a larger amount of dust in the air, the actual count was the minimum and not the exact figure.

He had noticed that Dr. Mavrogordato had referred to the old standards being based on light-ground counts. He (the speaker) was not quite certain whether there was not some misunderstanding on that point. He understood that the standard of 300 particles per c.c. which was adopted by the Miners' Phthisis Prevention Committee was not based on a count at all. It was based on the weight and average size of dust found in the lungs of men who had died of silicosis and on consideration of the amount of air breathed by a man in a certain number of years.

Dr. Mavrogordato: That is quite right, so far as I know.

Mr. C. J. Gray, continuing, said the standard of 300 particles would not indicate that air giving a count of 300 particles with the konimeter would be safe; if the konimeter could only show 20 per cent. of the dust a count of 300 particles with the konimeter was five times as dangerous as the standard permitted by the Miners' Phthisis Prevention Committee. If they were to accept an arithmetical standard they must not assume conditions to be satisfactory if the konimeter count equalled that figure.

Another point to which Dr. Mavrogordato had referred was the fineness of dust. He thought they would all agree, from their observations, that the dust was finer than it was formerly: the coarser dust was less common in their konimeter slides. He thought the reason was that the use of water had been successful in dealing with the coarser dust. Now that they were using water everywhere and passing it through machine drills the dust contained very few coarse particles. He did not think that the change was due to any change in the character of the rock, or anything of that sort.

He agreed with the suggestion that they should standardise methods with regard to konimeter work. It was perfectly true, as he had pointed out before, that if they did not have the same condensers and the same illumination, they would have different counts, and, if they were to have comparable results from different observers, they must have similar methods of working; but he did not think they should put any limit on the use of varying methods in research. If, for instance, the methods used some years ago had become rigidly standardised, they would have lost a great deal of knowledge which they had at the present time. He would be very sorry, if, for instance, he and his assistants were given an instruction that they must not use powers on the microscope different from some particular standard or any instruction of that kind. They must always do a great deal of work with the different methods in order to increase their knowledge.

In the past most of the work had been in the nature of research; it had not been simply a matter of putting down figures on a standard basis. In the Mines Department especially they had always been trying to get further information, and they would still have to go forward in their research methods. They did not really care in most of their work whether their results were comparable

as long as they ascertained what the conditions actually were in any particular place. That was what they were driving at rather than the getting of results which might be comparable with somebody else's, both being equally short of the full truth. Of course, it was for their own information that they were working in that way, and there was work in which it was most desirable that results should be comparable. He had certainly greatly enjoyed what he had heard that evening, and he was sure the rest of the members present had done likewise.

Mr. H. A. White, speaking as a layman, said he would like Dr. Mavrogordato to make it perfectly clear to them whether, in the Kotzé konimeter, they were dealing with a shunt method which gave them a definite percentage of the dust which was passing, in the same way as in their larger electrical apparatus they did not pass the whole of the current through the ammeter, but a definite proportion—whether they were dealing with a shunt method and a constant percentage, or whether they were dealing with an inaccurate method with an average efficiency of seventeen per cent., which with a normal variation would give them efficiencies varying from 4 to 68 per cent. It was obvious in the latter case and in comparison with the figures given them this could not be of much use in the furtherance of their ultimate object, which was to eliminate the dangerous dust from the mines.

Mr. A. R. H. Flugge-de Smidt said he would like to ask Dr. Mavrogordato how the table was obtained comparing the number of particles per c.c. of a certain average diameter with the weight.

Dr. Mavrogordato, in replying to the discussion, said he was in the unfortunate position of agreeing with everything that Mr. Gray had said. It was an unfortunate position because there was much more advance in knowledge by disagreement than agreement, on occasions like this. The first point Mr. Gray raised was the question of standards of 100 and 300. Mr. Gray and he were talking about different things. He (the speaker) was sorry he did not make himself clear. He was not referring to the standard of 300 particles per c.c. which had been suggested as the ideal at which to aim; what he was referring to was Dr. Moir's work represented in the table thrown on the screen, in which certain numbers of particles

per c.c. were supposed to be related to certain gravimetric determinations varying according to what was the mean diameter of the particles; that was all he was referring to when he spoke of an average of 100 particles per c.c. corresponding to a milligramme per c. metre. That was determined, he thought, by Dr. Moir by light-ground counting. As they would see, Dr. Moir's returns varied from between 30 to 140 in laboratory work, and the Chamber's varied between 70 and 150 in practice. That was all he was referring to when he talked about light-ground counting. He was not referring to the 300 question.

If he might skip for a moment to Mr. Flugge-de Smidt's question as to how the table was obtained—Dr. Moir applied the principle of soil grading analysis to air-borne dust underground; that was to say, he took a sample in water, let it settle for varying lengths of time, and then took it after varying times for settling and then sorted out his sample by a series of grading experiments; it was on that that he based his conclusion as to what was the mean size distribution in any average sample—that was to say what was the average number of particles above 5 microns—round about 3, and so on. He (the speaker) thought the two questions went together, those of Mr. Flugge-de Smidt and Mr. Gray. For himself, he thought that their ideal dust sampling method would be a gravimetric sample that had been washed and ignited and then subjected to grading analysis. If they could get a washed, ignited sample, they would then be working on phthisis-producing dust. It was much too elaborate for routine work. He knew Dr. Moir was working on something that would give them the best method in the world if he got it out.

As to the concentration. Mr. Gray had said 300 particles per c.c., but his (the speaker's) reference was to the relationship between mg. per c. metre and particles per c.c. As to the argument of the amount of dust a man will inhale in a lifetime, that was an argument which meant nothing to him. First of all, there was no relation between the amount of the damage in the lung and the number of particles that were in there. They had to deal with what a man inhaled, and no two men would agree in that respect. The amount of dust that went in would vary with the amount of the man's breathing. If he was a boy doing hammer work,

for instance, or walking about the mine, the air that entered his chest underground would vary by hundreds per cent. They could not get any information by trying to get at the average number of particles a man would inhale in the course of a lifetime underground; the amount he breathed would vary with the type of work he was doing. That was one difficulty, and the other was that no two men would react in the same way to the dust they inhaled. How did they know then that their mining was a dangerous occupation? It was not because of the sugar tube, the konimeter, the katathermometer, or any other toy they liked; it was just practical experience.

They knew that in 1911 30 per cent. of the men employed underground had got phthisis. They knew that by 1916 something under 6 per cent. had got it. They knew now that something under 4 per cent. had got it—that was to say, they knew they had an occupational disease which they called miners' phthisis. They knew they had got less of it now than they used to get. They knew that in the early years if a man was going to get the disease he got it on the average in two years; they knew now, if he was going to get the disease, he got it on the average in something between 9 and 10 years. They knew the fall off in the number of men who got the disease and the increase in time they took in getting it was related to the fall in dust concentration as determined by their two methods. There was less miners' phthisis than there used to be. Their methods of testing the dust did give them the information they wanted, and the incidence of the disease had fallen off with the fall in the amount of dust.

He himself would not worry about 300 particles per c.c. based on any arithmetical argument. They were getting a measure with the konimeter and they had reason to believe, if they could get down to that number of particles as shown by the konimeter it was about the best that was obtainable under practical working conditions. He thought they could do better now than in 1916. They were merely relating dust as estimated by their standards with the amount of disease they got as found out by examining the workmen. That was the real standard they worked to, and not a theoretical arithmetical argument.

Then, as regards the sugar tube. He was entirely a disciple of Sir Robert Kotze's. He (the speaker) called it a filtration method

because that was the usual term. He thought it was an impinger, just like a konimeter was. He called the konimeter a precipitation method, but it was not solely a precipitation method; it was a condensation method as well. He simply used the standard methods of description. He would have been more nearly correct if he had called them "Condensation" and "Impinger" methods. He had simply used the ordinary term.

Again, he was entirely in accord with Mr. Gray in saying that research and routine were two different things. It was only suggested that for purely routine work a standard method should be adopted.

Then a question had been raised by Mr. White as to whether the konimeter was a shunt method. There were two things that had to be considered. The konimeter took a sample of 5 c.c. or 10 c.c. according to the type of konimeter they were using, and the two points were: (a) Did the konimeter catch a uniform proportion of the dust in the 5 c.c. that entered it; (b) did the dust in the 5 c.c. that entered the konimeter through the narrow orifice represent the dust in the air at large? On this point Sir Robert satisfied himself by using a konimeter that worked in relays; with up to about 500 particles his konimeter caught about 90 per cent. of the dust that entered in the 5 c.c. When the Chamber of Mines was working out the circular konimeter they repeated experiments on those lines with the relay type of konimeter—that was to say, the air hit one slide and then went on and hit others; they found an efficiency of over 80 per cent.

After an interval of many years, Dr. Owens, who was on Government service in England, and who had apparently never heard of the konimeter, invented one on his own account, and, curiously enough, he standardised it by precisely the same method. His figures were practically identical with those of Sir Robert's and the Chamber's. So, so far as that went, he thought they might say that with moderate concentrations of dust Dr. Owens' worked up to 700, and Sir Robert's and the Chamber's up to 500 and caught over 85 per cent.

As to whether the dust in the air that entered the konimeter represented the dust in the air at large they had the following tests: to begin with on laboratory determinations there should be a certain relation between particles per c.c. and mg. per c.m.,

and on the whole this correspondence obtained in practice. He had tried another test by counting the particles in gravimetric samples and comparing them with corresponding konimeter samples. The konimeter, by this test, caught about 21 per cent. of the particles caught by the sugar tube, but on the whole it varied uniformly with the sugar tube. Similar tests by American and Australian workers have given similar results. He agreed with Mr. White in regarding the konimeter as virtually a shunt method, but accepted the shunt, on the above evidence, as representing the conditions. He agreed with Mr. Gray that the concentration of dust in the 5 c.c. that entered the konimeter was less than the concentration in the air at large, but thought that his (the speaker's) figure of 21 per cent. was on the low side because the sugar tube counts were made on a naked slide, whereas the konimeter spot was in a film of adhesive, and films of adhesive lowered counts in his experience, which agrees with Dr. Owens' in this respect. Again our sugar tube samples are small, and the sugar tube stops a bigger proportion in the first two minutes or so than later, and small samples mean relatively high returns.

Mr. H. A. White said he was not quite clear as to the difference between the 20 and the 80 per cent.; he was not quite clear where the difference disappeared if they did not catch it in the konimeter.

Dr. Mavrogordato said supposing they put a tube over the konimeter orifice and blew some tobacco smoke into it they would see a large column of smoke going into tremendous vortices when the konimeter went off, especially where it came into the orifice of the tube. As a matter of fact, what they got was a little, straight, narrow column going straight up from the orifice of the konimeter and whirling vortices all round it, and they probably kept out a considerable proportion of the dust going into the konimeter by acting as air-veils. If they came to think of it, with whirling air, they had both gravity and a centrifugal force working, and many particles were below the average size, where gravity or centrifugal force worked freely on them. He thought the ability to catch hold of a particle and pull it in, especially with the smaller particles, was probably not quite sufficient. He thought Mr. Gray would agree with him

when he said that the greater they made the pace at which the air went into the konimeter the greater they got their count; the faster the air passed through the orifice the higher the counts. Dr. Owens, working with his 400 meter rate as compared with their 80, got counts about four times as high as they did.

Mr. S. de Smidt enquired as to the effect of the acid treatment in eliminating water soluble particles?

Dr. Mavrogordato replied if he were using a naked film, that was to say, taking his spot directly on a clean glass slide with no film at all, the efficiency he thought would be fairly high; the efficiency was not nearly so great when working with a film of adhesive. When working in wet conditions underground they could probably dispense with their adhesive altogether; but where they were working with dry air and were depending for their fixation on condensation, it was quite a different matter. Consequently it was desirable, in order to be on the safe side, to work with an adhesive of some kind. They found vaseline was perfectly useless if they were going to do anything in the way of a test by acid, and the glycerine jelly as originally used was quite satisfactory, but there was some danger that the glycerine jelly underground might run. That was why, he thought, Sir Robert Kotze had given it up. Mr. Grills, of Messrs. Hortors, had recommended to them clarified fish glue as used in a great many engraving processes, and they found a good many of the particles on fish glue films disappeared when exposed to acid vapour without upsetting the spot. He thought they had really taken a great deal more trouble over it than it was worth in trying to get a method of removing the particles without destroying the spot for counting. All they wanted to know was the fact as to whether there was a lot of particles which disappeared on treatment with water or acid vapour, or whether there were not; it did not matter much about counting them again. In the Owens' konimeter they went up to a litre of air and got a line in place of a spot of dust. A great deal of Owens' work had been microchemical. He (Dr. Owens) found that as the water evaporated streams of water ran off laterally from the line in which the soluble particles were contained and they were redeposited. If they were working without a film of adhesive he (the speaker) thought

they would get at the water soluble particles all right by acid vapour and then by applying a drop of water. As shown in the lantern slide just put on the screen these particles came out of solution again, of course, but in quite a different form to that in which they went in.

Mr. Flugge-de Smidt asked whether Dr. Mavrogordato would express an opinion on Mr. Boyd's experiments on the calcium sulphate spots.

Dr. Mavrogordato had no doubt if Mr. Boyd had used fish glue as a medium he would not have got it all dissolved; but if he had used the naked slide, he thought he would, but he could not dogmatise. If they took the actual routine work they got hundreds of thousands done a year; if they took two or three men's work for a few weeks, such as Dr. Moir's, Mr. Ray's, and his own, it was something in the neighbourhood of 100 instead of 100,000. There were not enough data. His opinion was Mr. Boyd would have got it all if he had a plain slide, and that he would not if he had used a film.

NOTES ON ORGANISATION FOR PROMOTING INDUSTRIAL EFFICIENCY.

By S. DE SMIDT.

(*Printed in Journal, August, 1922.*)

REPLY TO DISCUSSION.

While I did not attempt to deal with all the difficulties which might crop up in the introduction of such a scheme as dealt with in my paper, I see no good reason why such a scheme, with a little initiative and organisation, should not be as workable on the mines of the Rand as in the iron factory. I think some of the points in my paper have been missed by some of the contributors to the discussion. I will reiterate some of these points. The efficiency or industrial engineers in the factory at Pittsburgh, referred to in my paper, do not work under the various foremen. They have no authority and cannot interfere with the officials. If they do they do it at the risk of dismissal. They work under the direction of, and are directly responsible to, the assistant general man-

ager. They make suggestions to the manager, and sometimes they effect improvements by working in conjunction with the foremen.

Mr. Chilton's point that it is not so easy to apply efficiency methods to underground work because the workmen are not directly under the eye of the foremen is all the more reason why more attention should be paid to plant, equipment and labour-saving devices. It is just here that the efficiency engineer will find his greatest field. Some of us in days gone by have taken some trouble in teaching natives how to lash stuff by actually taking a shovel and showing them how to do it, but we soon found what a hopeless task it was, as it would need an army of trainers to make any impression on the ever-changing native. It is therefore all the more necessary to devote more attention to mechanical means of helping efficiency by introducing labour-saving devices.

Mr. Pirow drew attention to other methods of cleaning out stopes and development. When I was in America I saw tunnels being cleaned out by compressed air shovels, and the rapidity and smoothness with which the work was being done amazed me. It is only now that mechanical shovellers are being introduced on the Reef—I understand that a certain mine has two on order. But why must we always imitate other countries in these advanced methods? Why should we not lead the way?

Mr. Chilton remarks that there should be trained observers before the introduction of efficiency engineers, but these efficiency engineers will be the ideal observers. Before a man can make suggestions for improvements that will be of any value he must have learnt to observe. To get the best advantage from observation you want highly trained and educated men with initiative, and not men who will do the work like automatons.

I think we are all agreed that the word "efficiency" used in connection with the sampling incident related by Mr. Wartenweiler should be replaced by the word "stupidity" or some other stronger term.

I have not said anything about safety and health in my paper, but safety and health are inseparably connected with efficiency. As a rule the efficient mine is the safe and healthy mine. A list of mines in order of their efficiency—not working costs—would correspond very closely with a list of mines in order of merit as regards safety and health conditions. Where is the mine that can get efficiency with poor ven-

tilation and health conditions, or with bad safety conditions, where sections of the mine are frequently disorganised by accidents, and where the underground officials spend a good deal of time attending accident inquiries and inquests?

SOME NOTES ON THE EVOLUTION OF THE KONIMETER.

By R. A. H. FLUGGE-DE SMIDT.

(Printed in Journal, November, 1922.)

DISCUSSION.

Mr. H. Pirow (contributed): In conducting tests on the lines indicated by Mr. Gray in his contribution to this discussion at the March meeting I have found that with particles of ordinary size, i.e., varying from 3μ to $\frac{1}{2}\mu$, proper adjustment of illumination enables me to obtain approximately the same counts with light and dark background.

With particles $\frac{1}{2}\mu$ and less in size I cannot claim the same results with the $\frac{2}{3}$ objective and 5 ($\times 12$) eyepiece adopted as standard for routine work by the Department. My counts vary from 50 to 100 per cent. between the light and dark background, the light background count being the less. The smaller particles can, however, be seen distinctly with light background by increasing the magnification.

With an average size of particle varying from 3μ to 1μ —the vast majority of konimeter spots I have examined fall within these limits—I claim that Mr. Gray's contention holds good.

I should like to associate myself with Mr. S. de Smidt's remarks regarding acid treatment. Practically all spots taken by me during the last year were treated with hydrochloric acid, glycerine jelly being used as adhesive. I have found reductions in counts up to 50 per cent. of the total number of particles present in the untreated slide. These reductions took place within 15 minutes of applying the acid. On re-treating the remaining particles with acid, and even after dropping acid directly on to the konimeter spot, no material reduction in the count took place. It is therefore difficult to understand how the photographs seen by Mr. F. de Smidt warrant the assertion that experiments carried out by Mr. Boyd "leave a big doubt whether all particles of

CaSO_4 are eliminated by treatment with HCl fumes." Those of us who dabble in photography have often unwittingly produced spirit photographs only to find that under the acid test of scientific investigation the spirits are banished again to their proper sphere. Personally I am quite satisfied that the acid treatment of konimeter spots attains the object it was designed for: to get rid of soluble particles likely to cause an appreciable increase in the count.

Finally I feel that Mr. Flugge-de Smidt's remarks at the March meeting should not remain unanswered. I have frequently attended the tests referred to by him, and, taking two to four spots with the Kotze konimeter to every one taken with the circular konimeter, have had no difficulty in counting them in an afternoon. Other officials of the department have done likewise. I am prepared to take the 400 spots referred to by Mr. de Smidt and count them in a day if it is necessary by these means to demonstrate that the Kotze konimeter can do the work. By taking 12 spots on a slide—as I have done during the last year—the 12 slides usually carried with the Kotze konimeter enable one to take 144 spots, or twice the number obtainable with the circular konimeter. The number of spots that can be taken is only dependent on the number of slides carried, and is surely too trivial a point to be considered a disadvantage of any type of konimeter.

OBSERVATIONS ON THE TIN-ANTIMONY-LEAD ALLOYS.

By R. W. IRWIN.

(Printed in *Journal*, March, 1923.)

DISCUSSION.

Mr. R. A. Cooper (contributed): I should like to thank Mr. Irwin for bringing forward these "Observations." His method of attack, consisting of a preliminary investigation of the properties of the various individual components of the alloys, seems to be very logical.

I think that Mr. Irwin's note will be rendered much more valuable, however, if he will give us some conclusions drawn from his tests, as they seem to be open to a great deal of misinterpretation. For instance,

with the typical lead base and tin base alloys, the metals are compressed by the testing machine to a considerably greater degree when cast into a mould at 150°C . than when the mould is at 20°C . The same thing applies to the chilled lead-antimony eutectic, and the author definitely states that the ideal structure to withstand compression is obtained in the quickly cooled alloy. Nevertheless it is generally advised by authorities that the mould should be heated to about 100°C . (see Law's "Alloys," page 245, Gulliver's "Metallic Alloys," page 376, etc.). The conclusion likely to be drawn from the figures published is that the cold cast metal in these cases is superior. Are we to take it that in some cases the "compressibility" and the "practical value" are in inverse ratio?

Of course slow cooling of thick sections of some white metals results in segregation by flotation of antimony-tin crystals, and this is to be avoided by a regulated rate of cooling.

It would be unwise to place too much importance on the tests of tin-lead and antimony-tin solid solutions, as these were not in equilibrium, and results may be vitally affected by the presence of a second phase. The presence of the "hard constituent" noted by the author is a fatal objection to considering these specimens as acting like pure solid solutions. The antimony-lead eutectic poured into a mould at 150°C . and then chilled in water may be compared with the first two typical alloys. It naturally resists compression better than a more slowly cooled metal.

It is difficult to understand the author's finding that "the crystallisation of the predominant partner is more marked in the rapidly cooled eutectics," and that ternary alloys have a more homogenous structure when slowly cooled. It is more usual to find crystallisation and segregation suppressed by rapid cooling and assisted by slow cooling.

Possibly the casting temperature may have had a much greater effect on the cooling rate and structure of the metals than the mould temperature had, and it would be advisable to control both very closely for research purposes. The "compressibility" test as carried out with the dropping hammer should be an excellent guide to the selection of a metal suitable for bearings where resistance to concussion is an essential, but it can be conceived that it may not give at all a true

indication of the properties of a metal intended to carry a steady load. A lead base metal, which is comparatively soft, has been found quite equal in service to a tin base metal, under prolonged and very carefully controlled test under working conditions in this country on railway wagons.

In conclusion I would like to suggest to Mr. Irwin that as he has initiated this research he himself is undoubtedly the right person to carry it further on similar lines, and I hope that he will give us another note on the subject when he has advanced it another stage.

Mr. W. H. Sharpe (contributed): In connection with the paper read by Mr. Irwin the following note may be of interest to the Society:—

The South African Railway Administration's tests on the 11in. x 5½in. journals of 50-ton trucks have been running for over three years, but the tests are not yet complete. The metals under test are:—

- (1) 83% tin, 6% copper, 11% antimony.
- (2) 60% tin, 6% copper, 12% antimony, and 22% lead.
- (3) 5% tin, 2% zinc, 14% antimony, and 79% lead.
- (4) 20% antimony, 80% lead.
- (5) 5% tin, 15% antimony, 80% lead.

Results from No. (2) have been disappointing in these tests as it was thought this mixture would give results comparable with results from No. (1), but average figures of wear indicate that (2), (3) and (4) give similar results, these being 70% greater than results from No. (1). It must be remembered that the amount of wear per bearing in any case is very small.

Results so far indicate that metallic contact between bearing metal and gun-metal shell is superior in the case of the lead-base metals. This result was not anticipated. It is considered that No. (5) will ultimately give the best results.

In these lead-base metals the prevention of segregation is most important, and this result can only be achieved by continual stirring of the metal in the pot and by pouring at the proper temperature so that the time of cooling will not be prolonged, the chill and gun-metal shell being heated only to ensure the absence of moisture.

It may interest your members generally to know that on a 5½in. diameter journal an arc of contact having a 2½in. chord shows no appreciably greater wear than one having a 3½in. chord; it would appear that

on a lubricated bearing the principal factor producing wear is the starting of the bearing from rest plus the ensuing period up to a stable condition.

All the bearings are lubricated by means of a wool waste pad under the journal; they are examined every six months, and between these periods the axle-boxes are sealed.

Any criticism which your members may make regarding the disappointing results with No. (2) and regarding Nos. (4) and (5), with particular reference to the tin content of No. (5), will be very much appreciated.

BOOK REVIEW.

"ELEMENTS OF OPTICAL MINERALOGY: AN INTRODUCTION TO MICROSCOPIC PETROGRAPHY," by N. H. and A. N. Winchell. Second Edition. Part I., Principles and Methods. New York: John Wiley and Sons, 1922. London: Chapman and Hall.

Students of petrography are well provided to-day with good introductory books, which, together with good teaching and facilities for laboratory work, account for the greater definiteness and accuracy of most references to the mineral components of rocks met with in geological papers written within the last ten years as compared with those of 25 years ago. The brothers Winchell's book is a new edition of the first part of a well-known text-book which has been to a very considerable extent rewritten. Emphasis is laid on refractive indices as the most important character for the determination of minerals in powder as contrasted with their condition in thin sections of rock, and the authors do well in insisting on the propriety of examining rocks in powdered form as well as in thin sections. For many practical purposes rock powders give sufficient information, and they are prepared in a minute, while thin sections take a considerable time to make, and the appliances are cumbersome. Methods for the determination of refractive indices under the microscope have long been known, but they have been too little used owing to the fear of the trouble of getting trustworthy results; even in this book enough stress is not laid on the importance of facilities for determining the indices of the liquids employed accurately and quickly if much use is to be made of the immersion method. The book is designed

for the needs of certain classes, and for general use parts of it are not very suitable; the chapter on chemistry, for instance, seems either superfluous or inadequate; a "normal salt" is defined on one page while the use of a "normal and neutral" solution of a salt is advised shortly afterwards without further explanation. A picture of Federow's universal stage is given on page 60, while no mention is made of its use beyond a very brief statement on page 176, where it is also said, and with justice, that such devices are too expensive for common employment. The appearances seen by the help of the microscope and ordinary, plane-polarized and convergent polarized light are clearly explained, and sets of observations are outlined at the close of chapters which should be of material use to students. The systematic description of mineral species will follow in subsequent parts of the book, but important rock-forming minerals are mentioned here in connection with appropriate subjects; the very useful diagram of the optical properties of the plagioclase feldspars on page 207 deserves most careful study, but further explanation in the text would have been advisable.

A.W.R.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

FUSIBILITY OF ASH FROM COALS OF THE UNITED STATES.—The Bureau of Mines has made a general survey of the "fusing" or "softening" temperatures of the ash from coals in the United States. Data on the fusibility of coal ash should be of considerable value to the coal consumer, mainly in connection with the troublesome formation of clinker resulting from the melting of the ash constituents in burning coal. When used together with the large number of coal analyses published by the Bureau, it will assist the consumer in comparing different coals and in selecting the coal best adapted to his purpose. The ash is the incombustible residue after complete combustion, and is derived from the inorganic mineral constituents. These are: (1) Inherent impurities in intimate mixture with the coal substance; (2) impurities such as clay, shale, pyrite and calcite that are present as partings, layers, or nodules in the coal bed; and (3) fragments of roof or floor mixed with the coal in mining. Coal ash varies so widely in chemical composition that no typical composition can be given, but generally falls within the following limits: Silica, 40 to 60 per cent.; alumina, 20 to 35 per cent.; iron oxides, 5 to 25 per cent.; lime, 1 to 15 per cent.; magnesia, 0.5 to 4 per cent.; and alkalis, 1 to 4 per cent.

The fusibility depends on several factors, such as ratio of bases to silica present, the particular bases, and the percentage of alumina. For example, ash extremely high in silica or in bases is not readily fusible. Ash low in iron is usually so high in silica that it is not readily fusible. Ash of coals high in pyrite contains much iron, and is likely to form clinker, the iron converting to ferrous oxide which unites with silica to form easily fusible silicates. Some coal ash is so infusible that little clinker is formed, and ash slightly fusible often forms a porous, spongy clinker that does not obstruct flow of air through the fuel bed and is easily removed. Ash with a low fusing temperature, say, 2,100 deg. F., may become quite fluid and spread out in a thin sheet, obstructing flow of air and localising the heat in the fuel bed. In general, the softening temperature of ash from coals in the United States ranges from 1,900 deg. to 3,100 deg. F., and may be classed as: (1) refractory ashes, softening about 2,600 deg. F.; (2) ashes of medium fusibility, softening between 2,200 deg. and 2,600 deg. F.; and (3) easily fusible ashes, softening below 2,200 deg. F. Further details will be found in Bulletin 200 recently issued by the Bureau.—W. A. SELVIG and A. C. FIELDNER, *Journal Franklin Institute*, December, 1922, p. 833. (J.A.W.)

A TESTING DEVICE FOR WEIGHING GALVANISED COATINGS.—This is based on the action of hydrochloric acid on the galvanised coating, collecting and measuring the hydrogen evolved. A ring of tinned iron or nickel, made to resist the action of the acid, is shaped to fit the surface to be tested, or in the case of wire, a definite length is placed on a glass plate under the flat ring. The rings are fitted with three hole stoppers, one carrying the filling tube, which is provided with a glass stop cock, while the other holes carry exit tubes. To make a determination the ring is sealed to the surface to be tested with acid resisting cement. 30 ccs. of concentrated HCl aq. is then placed in the filling tube, and about 5 ccs. admitted to the ring, previously filled, as well as the burette and tubes, with water by the aid of a levelling bottle. When the liberation of gas ceases the apparatus is rid of gas by filling the ring and connecting tube with water and forcing all the gas into the burette. The volume of hydrogen evolved in ccs. at 20 deg. C. multiplied by the factor provided with each ring gives the ounces of spelter coating per square foot of actual surface on one side of the culvert.—ANON, *Chem. and Met. Engineering*, Vol. 27, No. 16, October 18, 1922, page 797. (J.A.W.)

THE DETECTION OF NITRITES.—The following reactions serve for the detection of nitrites:—

(1) Addition of sodium thiosulphate solution, of about 0.5 N concentration, drop by drop to an alkali nitrite solution acidified with sulphuric acid yields a transitory yellow colouration, more or less intense according to the proportion of nitrite present; this reaction is perceptible with 0.0001 N nitrite solution.

(2) If a litre of water containing 0.0001% sodium nitrite is treated with about 2% of sulphuric acid and then with excess of thiocyanate and about 0.1 gm. of ferrous sulphate

an orange colouration is readily developed; if only 0.00001% of the nitrite is present the colour develops more slowly.

(3) Excess of aniline phosphate, either solid or in freshly prepared solution, gives a yellowish colouration, turning to reddish, with a 0.1% sodium nitrite solution; the reaction is not detectable with a nitrite solution of greater dilution than 0.001%. Benzene removes the colouring matter from the liquid, and the colouration may be distinguished from that given by hydrogen peroxide by means of sulphuric acid, which turns it red or pink.—P. FALCIOLA, *Chem. Trade Journal*, Gazz. Chim Ital., 1922, 52 II., 87-89, December 29th, 1922, p. 800. (R.A.C.)

A MODIFIED METHYL-ORANGE INDICATOR.—

A great many substances have been employed as indicators for acid-alkali titrations, but, in spite of the number available, only a few indicators, such as phenolphthalein, methyl-orange and methyl-red, are in favour for ordinary analytical work. The use of methyl-orange is almost universal, although the colour change from yellow through orange to pink is very difficult to detect in its initial stages. The authors of the present paper have sought to improve the colour change of methyl-orange indicator without altering its chemical properties. As the eye is most sensitive to change of hue when that change occurs to a neutral-coloured object, methyl-orange must be modified so that the colour change is from white or neutral grey to a shade in which one or more colours predominate to effect an improvement. A screening dye having a single absorption band in yellow-orange is added to the indicator solution so that when the latter is alkaline the colours transmitted are extreme red, green, and a trace of blue; the luminosity of the red and blue is small, so that the solution appears green. On acidifying, the methyl-orange band shifts into the green, reducing the luminosity of the latter until the red and blue preponderate, giving the solution a magenta colour. With the choice of a suitable screening dye there is a period during the acidification when the amounts of red, green, and blue transmitted are visually equal, so that the solution appears white, or neutral grey. The colour change is therefore green—neutral grey—magenta, that is, from one colour, through white, to a complementary colour; a change to which the eye is most sensitive.

Furthermore, the "grey" is a definite stage and can be recognised without previous experience, and repeated from titration to titration without the use of comparison solutions.

The screening dye finally adopted was xylene cyanole FF, the mixed indicator containing one part of methyl-orange to 1.4 parts of cyanole dissolved in 500 parts of 50 per cent. alcohol. The "grey" neutral point corresponds with a hydrogen-ion concentration of $\text{pH} = 3.8$.

Numerous experiments are detailed showing that the discrepancy between the mixed indicator and methyl-orange is almost negligible with N/10 solutions and stronger, and is non-existent if the usual salmon-pink is chosen as the methyl-orange end point. It was hoped that filtering screens for artificial light could be used to enable methyl-orange titrations to

be done at night-time. It was found that the eye adjusted itself to the general illumination, the colour of the indicator remaining substantially the same. Excellent results were obtained by using the modified indicator in conjunction with a 100-watt gas-filled "daylight" lamp with a blue glass bulb.—HICKMAN and LINSTEAD, *Journ. Chem. Soc.*, November, 1922, p. 2502. (R.A.C.)

MINING.

MINERS' WORM DISEASE.—A correspondent of the *Times* at Suva (Fiji), states that an important advance in regard to the treatment of ankylostomiasis has been made in the Suva Hospital by the use of carbon tetrachloride. Dr. Hall, Senior Zoologist of the United States Government, conducted tests for some months, and found that this substance was a perfect vermifuge for hookworms and other intestinal parasites in dogs. In the Suva tests 98 per cent. of worms in human beings were removed with one dose, almost without symptoms. The treatment is still in the experimental stage, but is of great promise. The use of the new remedy will result in a great economy, as the cost is only an eighth of the old method.—*Colliery Guardian*, June 9, 1922. (J.C.)

PILLARLESS MINERS' LAMPS.—In a note on safety lamps, the Editor states that two types of pillarless electric lamps have been approved by the Mines Department of the Board of Trade. The objections to the pillar type are that they provide indifferent lighting and that the pillars cause alternate zones of darkness and light, which are irritating, if not actually harmful. The Mines Committee report that, after continued trials over 18 months under working conditions, no increase in breakage is reported for the pillarless types of lamps over the pillared types. The objection to the lack of protection to the eyes against upthrown light may be met by the provision of a movable reflector attached to the handle of the lamp.—EDITOR, *B.E.A.M.A. Notes and News*, Vol. XI., No. 3, September, 1922, p. 585. (H.P.)

CHANGES OF ADDRESS.

BEATTY, J. W. S., *1/o* Vogelfontein: P.O. Box 13, East Rand, Transvaal.

DEWAR, T. N., *1/o* Boksburg: P.O. Box 183, Brakpan, Transvaal.

GALBREATH, N. M., *1/o* Barranquilla, Colombia, S. America: 32, St. Vincent Crescent, Glasgow, Scotland. (Until September, 1923.)

GORDON, W. S., *1/o* Village Main Reef G. M. Co. Ltd.: Village Deep, Ltd., P.O. Box 1064, Johannesburg.

RAMSAY, R., *1/o* Princess Estate & G. M. Co., Ltd.: Modder Deep Levels, Ltd., P.O. Box 326, Benoni, Transvaal.

TRAVERS, J. W., *1/o* Knight Central, Ltd.: New Rietkuil Mines, P.O. Box 7, Klerksdorp, Transvaal.

WRAITH, C. O., *1/o* B'wana, M'Kuba, N. Rhodesia: Gwaai Mine, P.O. Dett, S. Rhodesia.

THE JOURNAL
OF THE
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Vol. XXIII.

MAY, 1923.

No. 11.

Proceedings
AT
Ordinary General Meeting,
19th May, 1923.

The Ordinary General Meeting of the Society was held in the Assembly Hall, Scientific and Technical Club, 100 Fox Street, Johannesburg, on Saturday, the 19th May, 1923, at 8 p.m., Prof. G. A. Watermeyer (President) in the chair. There were also present:—

25 Members: Messrs. H. R. Adam, C. J. Gray, F. W. Watson, J. Chilton, R. A. Cooper, A. King, F. Wartenweiler, John Watson, H. A. White, J. A. Woodburn (Members of Council), J. A. Boyd, J. M. Dixon, E. C. Homersham, G. Melvill, P. T. Morrisby, W. W. Oliver, Lieut.-Col. E. Pam, F. D. Phillips, T. K. Prentice, J. F. Pyles, C. B. Simpson, R. A. H. Flugge-de Smidt, S. de Smidt, H. R. S. Wilkes and A. A. Wood.

4 Associates: Messrs. G. J. V. Clarence, R. Kethal, H. Rusden and G. C. Wilson.

8 Visitors and H. A. G. Jeffreys (Secretary).

MINUTES.

The President said, with their permission, he would defer confirmation of the Minutes of the meeting held on the 21st April until the next meeting. This was agreed to.

SCRUTINEERS FOR ANNUAL BALLOT.

The President drew the attention of members to the annual ballot for Officers of the Society, which would take place shortly, and called for nominations for scrutineers.

Mr. John Watson proposed Messrs. S. Newton, P. T. Morrisby, J. M. Dixon and G. Melvill.

This was seconded by Mr. C. J. Gray. There being no further nominations, the aforementioned gentlemen were declared duly elected.

NEW MEMBERS.

A ballot was taken for the election of new members, and the following gentlemen were declared duly elected:—

ELLIS, ELLISMERE, P.O. Box 15, Langlaagte Transvaal, Mining Student.

*FLETCHER, HAROLD C., P.O. Box 224, Bulawayo Mechanical Engineer.

*GORDON, G. B., P.O. Box 5184, Johannesburg, Consulting Engineer.

*OLSSON, C. H., P.O. Box 1871, Durban, Manufacturing Chemist.

*ROBINSON, VICTOR E., Rothamstead Mine, P.O. Box 34, Que Que, Rhodesia, Cyanide Manager.

*SPANDAW, H. J., P.O. Northrand, Transvaal, Analytical Chemist.

*WILLIAMS, G. W., P.O. Box 3404, Johannesburg, Reduction Shift Boss.

* Transfer from Associate Roll as from 1st July, 1923

The Secretary announced that the Council had admitted as an Associate Mr. E. D. Raine, Mining Engineer, P.O. Box 788, Johannesburg.

GENERAL BUSINESS.

ORDINARY GENERAL MEETINGS.

The President announced that at the last meeting of the Council it was resolved that, under the powers provided in the Constitution, it proposed to increase the number of Ordinary General Meetings during the year from eight to ten. The custom at present, as they knew, was to have two months' vacation at Christmas time and two months' vacation in July and August. The Council intended to reduce each vacation by one month; so that, instead of having only

eight Ordinary General Meetings per annum they would now have ten. The *Journal* would come out as usual during the other months. He would like to have the confirmation of the meeting of that resolution of the Council.

Mr. E. C. Homersham proposed that the action of the Council be approved.

Mr. J. M. Dixon seconded, and the proposition was agreed to.

The Chairman pointed out that, under the circumstances, their vacations would now merely be the month of July and the month of December; that was to say, there would be Ordinary General Meetings in August and January, or ten meetings in the year.

NOTES ON FAN VENTILATION IN DEEP MINES.

By J. A. WOODBURN.

From all that has evolved from the discussion on "Miner's Phthisis" it is very evident that improved ventilation throughout all the working places in a mine will have a very beneficial effect on the health conditions, and it is with the object of stimulating investigation and discussion on the most economical method of securing satisfactory distribution of the total air in a mine that the following notes are submitted.

Many fans are already installed on Rand mines capable of circulating large quantities of air, with water gauges up to six inches, and an examination of some of the problems relating to the behaviour of fans under varying conditions may be of interest.

In this connection it is well to realise that fans are somewhat like centrifugal pumps, which give their highest efficiency when designed for dealing with a definite quantity of water against a definite head or pressure.

With fans the same principle holds good, and if one is installed at a mine capable of passing, say, 200,000 cubic feet under a pressure or W.G. of, say, 4in., then it will be giving its highest efficiency when these conditions are fulfilled.

It may be, however, that to pass 200,000 cubic feet of air through the air passages in a mine, in addition to down and up the two shafts, will require considerably more than 4in. W.G., so the fan must be speeded up to increase the pressure, and loss of efficiency occurs. The increased H.P. is out of all proportion to the increased quantity or beneficial effect obtained in the mine by this increased pressure. On the other hand, should a larger quantity be allowed to pass by short circuiting, although the W.G. may be less than four inches and the total quantity of air be more than 200,000 cubic feet, the efficiency will drop

owing to the H.P. used up in the fan being in greater proportion than it ought to be.

In a comparatively shallow mine, say, down to 1,000 feet, and with several separate ventilation sections, a fairly extensive working can be efficiently ventilated and a large quantity of air circulated with a pressure or W.G. of three to four inches, but when mines are 3,000 to 4,000 feet in depth, and over, the W.G. required to pass a large quantity down and up the shafts alone may equal four inches, and much will depend on how many air distributing sections can be maintained underground, whether the ultimate total W.G. will be five or ten inches. Where the W.G. in any mine, that is the difference of pressure between the intake and return airway at any point is high, the risk of leakage is greatly increased, and as it is practically impossible to seal up all the stoppings and doors throughout the workings, the lower the W.G. or ventilating pressure at any point in a mine, the more easily will the air current be controlled.

This ideal of low W.G. throughout a mine might be accomplished in very exceptional cases by having five or six separate circulating sections, so that each section would take its required quantity or proportion at the same pressure or W.G.

As this happy condition can practically never be obtained, regulators have to be fixed in the shorter sections or in the sections where the pressure is less, to bring it up to the pressure of the longest section or the section taking the largest quantity.

In this way the section requiring the highest pressure fixes the underground W.G. and the regulators in the other sections only tend to increase the H.P. at the

fan, without any effective return; in other words, this increased H.P. is largely wasted.

There may be an objection to placing fans underground in collieries, especially in a fiery mine, where the breakdown of one fan might mean a dangerous accumulation of explosive gas. Auxiliary fans, however, have been placed underground in collieries with beneficial results.

This objection does not apply to Rand mines nor to metal mines generally, and it is worth considering the matter from the point of view of keeping the pressure or W.G. as low as possible.

If the surface fan had a capacity equal to the total quantity required, but working at a pressure of just sufficient to pass this quantity down and up each shaft, then fans in series might be installed underground throughout the mine to pass the required quantity from one to the other and so on to the bottom of the upcast shaft.

The following example is given as a purely theoretical one, to help to illustrate the principle, as practical application will vary according to the conditions of the different mines.

Assuming two shafts 3,000 feet deep, area of shafts 200 square feet, perimeter 70 feet, giving a rubbing surface of 210,000 square feet, and taking 200,000 cubic feet

Section.	Quantity.	Length.	Area.	Perimeter.	Rubbing Surface.
A	30,000 cu. ft.	5,000 ft.	80 sq. ft.	40 ft.	200,000 sq. ft.
B	40,000 cub. ft.	6,000 ft.	80 sq. ft.	40 ft.	240,000 sq. ft.
C	50,000 cub. ft.	7,000 ft.	80 sq. ft.	40 ft.	280,000 sq. ft.
D	80,000 cub. ft.	8,000 ft.	80 sq. ft.	40 ft.	320,000 sq. ft.

The velocity in each case would be:—

$$\begin{aligned} \text{Section A, } V &= \frac{30000}{80} = 375 \therefore v = .375 \text{ \& } v^2 = .14 \\ \text{.. B, } V &= \frac{40000}{80} = 500 \therefore v = .500 \text{ \& } v^2 = .25 \\ \text{.. C, } V &= \frac{50000}{80} = 625 \therefore v = .625 \text{ \& } v^2 = .39 \\ \text{.. D, } V &= \frac{80000}{80} = 1000 \therefore v = 1.000 \text{ \& } v^2 = 1.00 \end{aligned}$$

Applying the formula in each case the pressures will be as follows:—

$$\begin{aligned} \text{Section A, } p \times 80 &= .01 \times 200000 \times .14 \\ p &= \frac{280}{80} = 3.5 \text{ lbs.} \\ \text{B, } p \times 80 &= .01 \times 240000 \times .25 \\ p &= \frac{600}{80} = 7.5 \text{ lbs.} \end{aligned}$$

per min. of air as the quantity to be circulated.

The necessary velocity in the shafts would be 1,000 feet per min., and the velocity in thousands of feet per min. would be 1, which equals v , and v^2 would also equal 1.

The pressure required for each shaft would thus be, applying the formula $pa = ksv^2$, where

p = pressure lb. per sq. ft.

a = area sq. ft.

s = rubbing surface sq. ft.

v = velocity in 1,000 ft. per sec.

k = co-efficient of friction lb. per sq. ft.

and taking k as .01 per square foot per 1,000 feet per min.

$$p \times 200 = .01 \times 210000 \times 1.$$

$$2100$$

$$p = \frac{2100}{200} = 10.5 \text{ lb., or practically } 2'' \text{ W.G.}$$

For the two shafts this would be doubled, viz., 21lb., or 4" W.G.

If the whole of this 200,000 cubic feet of air had to be circulated throughout the mine in one long section, the pressure or W.G. would be absolutely prohibitive, so it is assumed the mine is divided into four separate sections of the following average dimensions, and taking the following quantities of air:—

$$\begin{aligned} \text{C, } p \times 80 &= .01 \times 280000 \times .39 \\ p &= \frac{1092}{80} = 13.65 \text{ lbs.} \\ \text{D, } p \times 80 &= .01 \times 320000 \times 1.0 \\ p &= \frac{3200}{80} = 40 \text{ lbs.} \end{aligned}$$

If there were only one fan on surface to

and C to bring the pressure up to that of D, and the fan would have to develop a pressure of 21lb. plus 40lb., or practically 12" W.G.

The H.P. developed in the air alone, apart from fan efficiency, would be—

$$\text{H.P.} = \frac{200,000 \times 61}{33,000} = 370 \text{ H.P.}$$

Under the suggested arrangement of having underground fans in series, the surface fan would only require to develop 4" W.G., sufficient for the two shafts, but with a capacity able to pass 200,000 cubic feet at this pressure. Then, if one fan were placed underground capable of passing the same total quantity at 13.65lb., and regulators were fixed in sections A and B the required quantities would pass in each of the three sections A, B and C, and approximately 80,000 cubic feet of air would be sent along to section D for about one-third of the distance, leaving two-thirds with a pressure of 40—13.65lb., or 26.35lb. still to be ventilated. Two smaller fans, each capable of passing 80,000 cubic feet, with a pressure of 13.18lb., placed about one-third and two-thirds distance, would carry the air throughout the whole section round to the upcast shaft.

Under these conditions the total H.P. in the air alone would be as follows:—

Surface fan	$\frac{200000 \times 21}{33000}$	= 128 H.P.
Large underground fan	$\frac{200000 \times 13.65}{33000}$	= 82.8
2 duplicate fans ...	$\frac{80000 \times 13.18 \times 2}{33000}$	= 64
Total		274.8 H.P.

By this arrangement the maximum W.G. in the mine at any point would only be a little over 2.5", whereas with the former arrangement the maximum W.G. in the mine would be about 7.5", and the H.P., excluding fan friction or efficiency, would be 274.8 H.P., as against 370 H.P., showing a saving of practically 100 H.P. on this example.

By placing extra fans underground there would be the capital cost of installing same, and the cost of upkeep; but against these there would be less pressure at any point

in the mine, and a greater certainty of getting the air where required, fewer regulators and less loss due to leakage throughout the system.

The figures given in the foregoing example are based on a co-efficient of friction, as already stated, of .01lb. per square foot of rubbing surface, per 1,000 feet per min., and as this co-efficient varies according to the nature of the airway and the irregularities of same the pressures and H.P. given may be on the high side, but the relation between the different methods remains the same, and exemplifies the importance of splitting the airways into as many sections as possible.

Various co-efficients are given by different authorities ranging from .005lb. to .02lb., and this figure of .01lb. has been chosen as being approximately the mean, and also because it is much easier to work with.

In dealing with regulators the following simple practical formula was found in an old note book, and as it is easy to remember and to work out it is given as a rapid method of arriving at the area of the necessary opening in the regulator to pass a given quantity of air with a definite W.G. or pressure, or to find what is termed the equivalent orifice of a mine.

Area of equivalent orifice:

$$.0004 \times Q \text{ in ft. per min.}$$

$$\sqrt{\text{W. G.}}$$

It is based on 0.65 as the *vena contracta* for air passing through a small opening, and looks less formidable than the formula given in Sir R. A. S. Redmayne's book (page 177) on "Ventilation of Mines," which is—

volume of air per sec.

$$a = \frac{0.65}{\sqrt{12}} \frac{2gh \times \text{density of cub. ft. water}}{\text{density of cub. ft. air}}$$

If we take an example, and use the simple formula given, say, 30,000 cubic feet of air to be passed through a regulator and 2" W.G. to be used up in so doing, then the area of the opening would be—

$$a = \frac{.0004 \times 30000}{\sqrt{2}} = 8.48 \text{ sq. ft.}$$

To prove the correctness of this formula it may be worked out thus:

$$v^2 = 2gh, \text{ or } v = \sqrt{64 \times h.}$$

h. being the height of air column corresponding to the pressure. One cubic foot of air weighs 0.0766 lb., therefore 13 ft. of air column will equal one pound pressure per square foot.

1" W.G. equals 5.2 lb., so that $2 \times 5.2 \times 13$ equals 135.2 ft. of air column corresponding to 2" W.G., velocity in feet per second therefore equals $\sqrt{64 \times 135.2}$ or 93 ft. p. sec. Velocity in feet per minute is 93×60 , or 5580, which is the theoretical velocity due to this height of air column. This figure has to be multiplied by 0.65, the *vena contracta*, which gives 3,627 feet per min. as the actual

volume in ft. per sec.

$$a = \frac{0.65 \sqrt{\frac{2gh}{12}} \times \frac{\text{density of water}}{\text{density of air}}}{\text{volume in ft. per min.}} \quad (h = \text{W. G.})$$

$$\text{or } a = \frac{60 \times 0.65 \sqrt{\frac{64 \text{ W.G.}}{12}} \times \frac{62.5}{0.0766}}{\text{quantity in ft. per min.}}$$

$$\therefore a = \frac{2574 \sqrt{\text{W.G.}}}{\text{quantity in ft. per min.}} \quad \text{or} \quad \frac{90.324 \sqrt{\frac{d_1}{\text{W.G.}}}}{d_2}$$

$$\text{or } a = \frac{0.039 \times Q}{\sqrt{\text{W.G.}}}$$

The area of orifice of passage of a fan may be worked out by the same formula after the W.G. for fan alone is arrived at experimentally. For instance, if a fan is driven at normal speed and the fan drift closed entirely, and the W.G. noted, which may be, say, 3" at this speed. If the regulator in the fan drift is then opened to, say, about 30 square feet area, which might represent the equivalent orifice of a mine, and the W.G. and quantity are noted, say, 2.5" and 120,000 cubic feet. The orifice of passage of fan will be got by taking this quantity of 120,000 cubic feet and the W.G. at 0.5" the difference between the reading when the fan drift was open and closed. If Om represents the equivalent orifice of the mine, and Of the orifice of passage of the fan, then

$$\text{Om} = \frac{0.0039 \times 120000}{\sqrt{2.5}} = 29 \text{ sq. ft.}$$

$$\text{and Of} = \frac{0.0039 \times 120000}{\sqrt{0.5}} = 67 \text{ sq. ft.}$$

velocity, and by multiplying by 8.48, the area obtained by the previous formula, 30,756 cubic feet is obtained, as compared with 30,000 cubic feet, the figure originally taken. This shows that the simple formula given is sufficiently accurate for all practical purposes. To be more accurate it might be written

$$\frac{0.0039 \times Q}{\sqrt{\text{W.G.}}} \text{ in cub. ft. per min.}$$

If the formula previously given from Redman's book be examined it will be found to be practically the same, thus—

This gives a ratio of Om to Of of 1 to 2.3. It will be seen that if Of equals Om, the W.G. for each will be the same and the efficiency due to the passage of air alone, apart from mechanical efficiency, could not be more than 50 per cent. If Of is double Om, then the W.G. will be as 1/4 to 1, and if Of be three times Om the W.G. will be as 1/9 to 1. It is therefore important that the orifice of passage of a fan obtained in this manner should be at least 2.5 times that of the mine, or portion of a mine from which it is exhausting the air.

In connection with the benefit of auxiliary fans in a colliery, Mr. Dron, at a meeting of the Mining Institute of Scotland, in October, 1921, said: "He had personally experienced a case where one section of a mine was badly ventilated. The distance from the face of that section to the pit bottom was about a mile and a half, and the other sections were comparatively near the

¹ Trans. M.E., Vol. LXII., page 88.

pit bottom. The fan speed had been increased and the W.G. increased, but it was impossible to send sufficient air into this long section. A small auxiliary fan was installed underground, with the result that the whole difficulty was removed."

There is also a very important example of the beneficial effect of an auxiliary fan underground working in series, or tandem, with the surface fan, at the Morro Velho Mine, in Brazil.

Here a cooling plant was installed to reduce the temperature and humidity of the air going into the mine, and at the same time a duplicate fan to the surface fan was installed at about 4,000 feet depth. A considerable improvement has thus been obtained, but how much is due to the cooling of the atmosphere at the surface and how much to the auxiliary fan at the fourteenth level, it is rather difficult to determine, as both installations started practically at the same time.

The depth of the mine is well over 6,000 feet, and the auxiliary fan has had the effect of not only increasing the total quantity, but of forcing more air to the lower levels without increasing the W.G. beyond the previous maximum at any point in the mine. Each fan develops fully 8" W.G., and by the time the air has reached the fourteenth level the W.G. has fallen to about 3", which is brought up again to 8" by the auxiliary fan. Mr. Eric Davies' in describing the installation, after showing the great benefit which had been obtained, says: "The cooling plant cannot, of course, claim all the credit for this enormous improvement, for cooling power as measured by the katathermometer is dependent on air velocity as well as temperature, and the velocity in the stopes, in common with that throughout the mine, is now 40 per cent. higher than it used to be, for reasons already given." It is very interesting to note that reliance was not placed on the cooling plant alone, but that it was deemed necessary to instal the auxiliary fan as well.

In contributing to the discussion on Mr. Davies's paper, Mr. E. Seymour Wood says:² "In my experience water gauge pressure between the intake and return airways shows considerable reduction from the bottom of the downcast shaft along the main intake roads to the workings. In some cases with a W.G. of 3" at the shaft bottom, at

a distance of a mile and a half from the shaft, the W.G. when recorded at that point was $\frac{3}{4}$ ", and at two miles it was scarcely $\frac{1}{4}$ ", and in some cases there was scarcely any W.G. at all, and consequently there was a reduction in the quantity of air and in the air velocity. If (he says) at or about this point, or further in the workings, a subsidiary fan was established, either to act as a forcing fan to push the air round the working places, or as an exhaust fan, it was found that the fan acted as a 'helper up' in drawing an increased quantity of air along the main intake road, setting up a W.G. considerably increasing the velocity and volume of air circulating in the working places, and reducing the temperature. By this system districts in hot and gassy seams, which had previously been unworkable, were worked in comfort. Subsidiary fans installed for this purpose have given as much as 30,000 cubic feet of air with a 1" to 1.5".

Where fans are run in parallel, say, two fans drawing air from the same shaft so as to increase the total quantity, it is essential that they should both be developing the same W.G., or if one creates a higher W.G. it will tend to draw air through the other and no benefit will be obtained from the extra fan running. Mr. James Black, in discussing a paper by Mr. Joseph Parker, on "The Operation of Fans in Parallel," says: "In concluding his paper Mr. Parker states that there is a decided field for usefulness for the application of fans running in parallel. I consider that statement to be far too optimistic, as it seems quite clear to me that the duplication of fans in parallel can be applied with advantage only when the internal resistance of the existing fan is abnormally large. Under normal conditions the use of two fans operating in parallel will result in a lower efficiency than can be obtained from one single fan."

Mr. Joseph Parker, in replying to the discussion, said: "I do not think I have much to quarrel with in Mr. Black's contribution to the discussion. I have made experiments with the fans at Wellesley Colliery in order to obtain more light on the operation of fans in parallel. I commenced by running the Waddle fan at a speed of 125 revolutions per minute. The throttle valve of the engine was marked, with the object of having it brought back to the same position in a later test with the fans in parallel. The volume measured was 131,000 cubic feet per minute, and the W.G. was 4.2 inches.

¹ Trans. M.E., Vol. LXII., page 333.

² Trans. M.E., Vol. LXIII., page 339.

The Sirocco fan was then run up to a speed of 172 revolutions per minute by adjusting the throttle valve, and it passed 120,000 cubic feet of air per minute at a W.G. of 4.12 inches. The Waddle fan was then restarted and put in parallel with the Sirocco fan, and the throttle valve of the engine driving the Waddle fan was put approximately into the position occupied in the first test. It was hoped that under these conditions the engines would speed up and develop as much power as when the fans were running singly. But this did not happen. The Waddle fan ran up to 133 revolutions per minute, and the Sirocco fan kept running at 172 revolutions per minute. The Waddle fan passed 136,000 cubic feet per minute, with a W.G. of 4.3 inches, while the Sirocco fan passed no air at all."

There may be a usefulness, however, in running fans in parallel if placed underground near the upcast shaft, and drawing air through separate sections of a mine. Each might be developing varying water gauges according to the quantities and lengths of the various sections, and in this way regulators might not be required or their effect would be minimised, as the actual W.G. at the bottom of the downcast shaft, or where the initial air was split, might be the same, and it appears to be well worthy of serious consideration and experiment to try and get efficient ventilation by means of a number of smaller fans throughout an extensive mine rather than depend on one fan at surface to deal with the total ventilating power. The effect of temperature and humidity would be considerably reduced by increased velocity at the working places where it is most required.

The splitting of the airways may be done throughout the larger portion of a mine, but there are always odd sections that have to be reached and there are development drives, winzes and raises, the faces of which are some distance from the main air course, and these form a problem of their own. Here auxiliary fans, or preferably ventilators of the Root's blower type, can be used carrying air through ventilating pipes.

It is essential that the outlet end of these pipes reach to a through ventilating airway, otherwise the air from the fan or blower will simply circulate back and forward in the drive and the real benefit of the ventilator will be lost. Iron pipes or rigid pipes are much better than canvas, as they can be used either for forcing or exhausting the air, whereas canvas pipes are only useful for

forcing or blowing the air, and much less air will be blown through several hundred feet of canvas pipe with the same fan or blower as through a similar length of rigid pipe, the friction being so much greater.

With regard to forcing or exhausting air through pipes from a development drive, the writer favours exhausting. By this means the foul air and dust produced at the face is carried away directly into the pipe, and even though the incoming air in the drive be moving at a low velocity yet it is fresher and better air.

If air be forced into the face then the foul air and dust is carried generally very slowly back along the drive, and unless a good velocity can be maintained, which is frequently not the case, then the air along the drive may be only diluted and take a long time before it is fully removed. Of course, in both cases it is presumed that there is sufficient pressure maintained to cause a good current of air to flow in the pipes.

The first principle of ventilation is that there should be two openings, an intake and a return, and where pipes are used exhausting from any place they should be finally connected with a main return airway. Ventilators of the Root's blower type are better where pipes are in use, as they develop a higher W.G. than small fans and the quantities, though small, are generally sufficient for several development ends.

Where small fans are used it may be that the friction of the air in the pipe is equal to, or more than the pressure which the fan produces and the result is that although the fan may be kept running it only compresses the air in the pipe, and maintains it at this pressure, and no current may be passing at all.

In long development drives, whether it will be more economical in the long run to fix in ventilating pipes with blowers or to make the drive wider and build a solid brick or concrete wall along the drive so as to get a larger ventilating airway, or to make a second drive in the footwall under the reef with through connections at intervals, will be a matter for each individual mine or each district in a mine to determine, and much can be done by anticipating the ventilating problem in laying out fresh development.

There is no question, however, but that attention to the proper directing of the air currents in a mine, even at considerable

cost for special air connections, pipes, blowers, etc., will lead to more economical work, less compressed air consumption and higher tonnage per man per shift.

In the Bisbee district of Arizona underground fans are in use ranging from 10,000 cubic feet to 200,000 cubic feet, and developing water gauges from one inch to five inches for the general distribution of the air in the various sections, and for development drives, raises and winzes small blowers driven by 2 h.p. motors are used to ventilate through ten-inch pipes.

Mr. Charles A. Mitke, mining engineer, in a book on "Standardization of Mining Methods," published in 1919, gives some interesting diagrams showing the benefit of improved ventilation on the general efficiency of the mine.

These diagrams give the results of two years' record and show very clearly an increased tonnage per man per day, a reduced consumption of compressed air, and a consequent considerably reduced cost per ton.

He gives a specification of what he calls "a good working atmosphere" as follows: Temperature, 78 degrees F.; humidity, 80 per cent.; velocity of air, 125 feet per min.; and volume per man, 350 cubic feet; and, he adds: "Although this may be more than can be obtained in many places, it is the ideal to be aimed at."

In dealing with blowers for development drives, he says: "The pipes should reach to the working face so that it will not be necessary to use compressed air. After the drift has progressed about 3,000 feet the blower should be provided with a reversing attachment in order that it may be reversed to clear the smoke away, and after that be run as a pressure fan to furnish good air to the heading."

Referring to the diagrams previously mentioned, Chart No. 4 illustrates the decrease in the quantity of compressed air used, owing to the installation of the ventilating system. The curve shows a drop from 110 millions cubic feet per month to 80 millions cubic feet per month. Chart No. 5 shows the increase in tons per man per shaft, which is from 2.5 to 7 tons; and Chart No. 6 illustrates the decrease in the cost per ton of mining the ore, which shows a drop from one dollar fifty-seven cents to eighty-five cents, or a decrease of nearly fifty per cent.

The writer has brought forward these extracts to show that actual experience has proved the economical value of having what is termed a good working atmosphere in all

mining places, and it is in the hope that some figures will be brought forward regarding quantities and water gauges of fans at present installed and how these quantities of air are distributed throughout the mines that these notes have been written.

The writer is confident that a careful investigation into this important matter will lead to better distribution, with probably less expenditure for power, though more expenditure for doors, stoppings and pipes, and the making of special air crossings and connections, which will be more than compensated for by increased efficiency later.

Mr. J. Chilton, in proposing a hearty vote of thanks to Mr. Woodburn for his very excellent paper, said that ventilation was a problem that was interesting to all mining engineers. In former years, when mines were comparatively shallow, ventilation was a very simple matter; but now, when it cost half-a-million pounds to put down a shaft and that shaft would only pass a certain quantity of air, ventilation became a serious problem. It was rather wonderful when they considered how fashions changed in regard to ventilation. There were some at that meeting who would remember the storm of opposition that arose when a fan was put down at the Village Deep some fifteen or sixteen years ago. The whole force of mining practice was against this novelty, and yet to-day they found there was hardly a mine on the Reef that had not fans underground. In the matter of ventilation they were certainly progressing.

He was very much interested to note Mr. Woodburn's remarks about the ventilation of drives, because that was the point where they could attack miners' phthisis; and it was the ventilation of drives that was the greatest problem. They could pass very large quantities of air through a shaft, but it was getting the air into the corners and nooks of the drives; that was the ventilation problem that faced most mining Engineers.

He did not want to go into a detailed criticism of Mr. Woodburn's paper that evening, because he believed it wanted deep consideration before launching out into that; but he was sure he echoed the sentiments of the meeting when he proposed a vote of thanks to Mr. Woodburn for his excellent paper.

Lt.-Col. E. Pam. in seconding the vote of thanks, said that he had read an advance copy of the paper by Mr. Woodburn, and

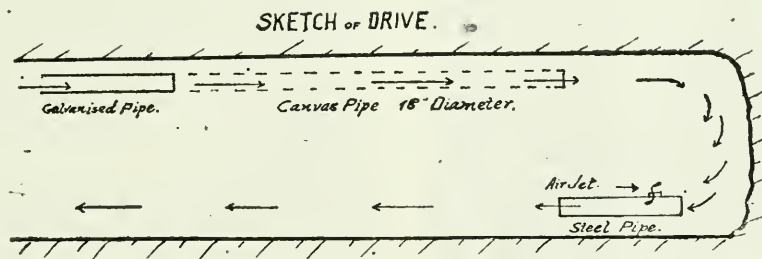
found it of great interest; but he was, nevertheless, a little disappointed. The author had taken up the general point of view and had not taken fully into consideration the difficulties with which they were met on the Rand. Mr. Chilton had drawn attention to these difficulties which Mr. Woodburn only mentioned in the form of a quotation from some American engineer. This engineer calmly said it was advisable to deliver the fresh air right up to the end, but did not explain how to do it. It was a difficulty, as a matter of fact, on which Mr. Chilton and the speaker had worked some time ago, and a very serious one from the point of view of miners' phthisis.

In most mines, especially on the Far East Rand, in big drives large ventilating pipes were used—not 10 inch., as quoted by Mr. Woodburn, but usually 18-inch or 24-inch, and either delivered fresh air or drew out from near the end of the dead-ends. That point of "near" was the trouble at the present moment, and, unfortunately, in nine

pipe and causing the fresh air to circulate right up to the face.

We had considerable trouble in keeping the scheme going, but it is possible, and only a matter of discipline.

After the strike, when it is undoubtedly more easy to get work done and to persuade the men to look after their own health and the health of their boys, it is possible to take a stronger line, and on quite a few of the mines now it is recognised that galvanised piping must come to within ten feet of the end. It means additional work to take down the pipe in 30 or 40 feet lengths and carry it back before and after blasting. Not at Modder East only, but on Brakpan and on other mines it was proved possible with not too much work to carry that pipe right up to the face—it might be five feet or ten feet back. If a thousand cubic feet was available ten feet back from the face, the dust was undoubtedly washed out and left the place quite clear.



out of ten dead-ends on the Rand to-day these pipes ended 100 to 150 feet from the face. It is clear that when delivering perhaps 1,000 cubic feet per minute through an 18-inch pipe, 150 feet from the face, the air will issue from the pipe and short-circuit back to the mine upcast as quickly as it can. In the majority of cases therefore, the fresh air did not reach the face where the boys and the white men were, and where dust was being made.

A year or so ago Mr. Chilton and the speaker used a canvas pipe from the end of the galvanised pipe 120 feet from the face to a point about 25 feet from the face and laid on the ground a steel pipe on the other side of the drive, reaching from about the machine bar to overlap the canvas by a few feet. In this pipe was a jet going in the opposite direction to the air in the main

There will be a certain amount of friction before everybody accepts this scheme, but not more so than in getting the water service more or less right. It is obvious, of course, on the Central Rand and on mines where the drives are very much narrower the difficulties will be very much greater. Where there is a brattice, the brattice naturally stops at least fifty feet from the face, as at West Springs, and some auxiliary galvanised piping is essential.

Mr. Woodburn's paper is undoubtedly of great interest; and the speaker would like to second a cordial vote of thanks on behalf of the Members.

The President said he would like to endorse what the proposer and seconder had said. A very valuable paper had been read before the Society, and one which would lead to a great deal of discussion.

THE RECOVERY OF GOLD ABSORBED IN REDUCTION WORKS.

By A. KING

When a mine is finally closed down and the reduction works cease to operate, certain rich deposits, which generally cannot be obtained when the plant is working, become available for treatment. Every part of the plant must be examined, particularly the ground outside and underneath the stamp-mill, tube-mill and plate-house buildings. In these examinations the panning dish becomes an exceedingly useful accessory in determining whether accumulations of material are likely to pay for treatment. Rubbish and scrap heaps outside these buildings should be closely examined; small quantities of rich sand or minute particles of amalgam may inadvertently have been carried out of the buildings on discarded material such as shoes, dies and liners, and due to weathering of the material or disturbance of the heap the ground underneath may have become so enriched that it will pay to remove the heap, knocking off any loosely adhering material and treating the ground underneath. Spillage from mill launders and leakages of pulp under stamp-mill floor and round mortar boxes themselves require thorough cleaning and chipping. The wooden floor of a stamp-mill building, in which amalgamation has been practised, should be pulled up and the sweepings from the boards collected. Most pulp launders will require chipping, and those in the tube-mill circuit should be burned and the ash collected; even concrete launders will re-pay chipping and examining for crevices and cracks, which should be opened up. Classifying and dewatering cones, feeders, screens, tube-mills, etc., should be chipped and the tube-mill liners will, of course, yield appreciable quantities of amalgam. A second rich chipping may in many cases be detached after sufficient drying and weathering of the wooden launders and steel plates. Stated briefly, it will be found that more or less rich deposits of scale, etc., have accumulated in many parts of the plant where pulp carrying amalgam or free gold used to flow. Deposits of fibre (from wood and sacking) accumulate in certain launders and cones, and these are invariably rich.

The most economical methods of collecting and treating these various deposits will depend largely on local conditions, on the

quantities to be treated and on what units of the reduction plant it may be permissible to keep in use. From an extraction point of view the most satisfactory method is to retain in use a unit of the crushing and amalgamating or concentrating plant along with the necessary cyanide treatment plant. It will generally be found that much of the material will require re-grinding in order that a satisfactory extraction of gold may be obtained, so that a tube-mill together with the necessary amalgamated plates or corduroy tables, a cone for classifying and returning the insufficiently ground material, and the necessary cyanide units, or some modification of that combination will be found most satisfactory if there be sufficient bulk of material to warrant such elaborate treatment.

In this regard it may be of historical interest to record that in the final clean-up of the Knights Deep Reductions Works during 1920-21 some 15,000 tons of ore from mill and shaft bins were treated by the above method with very satisfactory results, an all-slimed product being produced by the tube-mill.

In the event of re-grinding not being possible a combination of screening and sluicing or concentrating may be adopted in order to produce a product rich enough for barrelling or for sale. These methods may in any case be necessary with such material as ashes, wood chips, canvas, launders, etc., which have first to be burned to render the gold content available. Rich sand and scalings from mortar boxes, launders, cones, pipe-lines, etc., should be re-ground in barrels or a small tube-mill, amalgamated or concentrated, and the residue sold or treated by cyanide with air agitation. Accumulations of fine steel and battery chips should be examined; it will probably be found that exposure and oxidation have made it possible to separate by screening a product sufficiently rich for sale to smelter, and in handling this product a look-out should be kept for osmiridium.

The usual method of scaling amalgamated plates (see *Rand Metallurgical Practice*, Vol. 1., p. 84) gives satisfactory results, and the gold content of the copper may be economically reduced to under two pennyweights

per ton copper by repeating the operations twice over. Unless special precautions are taken, however, the plates buckle so badly that without considerable trouble and expense they cannot be brought into a fit state for re-use as amalgamated plates and are of further value only as scrap copper. If it be desired to re-use the plates either for amalgamation or as tables for corduroy concentration, they should be well supported during the heating operation; excessive heat must be avoided, and the heat of the fire underneath should be diffused as equally as possible all over the plate. The following arrangement was found to answer these requirements: 60lb. rails were supported at ends and centre about 18in. from the ground and about 2ft. apart, short pieces of old flanged pipe with the bottom ends sunk in the ground serving as supports; steel plate, three-eighths of an inch thick, was placed on the rails, forming a flat table for the copper plate. The width and length of the table were a few inches greater than the copper plates, and the whole was surrounded by loose sheets of iron to concentrate the heat from the fire underneath, which burned under the whole of the plate, but the heat from which was diffused through the steel plate so that very little buckling took place. At the first heating the temperature was only sufficient to drive off the mercury, and the second heating was continued until the copper oxide scale was separating from the plate. The latter was then carried on to another even surface (built up from old tube-mill-liner bars) and inclined towards a suitable wooden box into which the scale was washed by water from a hose. The plates had, of course, been prepared for the scale-forming operation by the chemicals recommended in the above reference. The small amount of buckling which took place during heating disappeared during cooling, and the collection of the scale by this means was found convenient and safe. The heat treatment appeared to soften the copper so that re-setting with mercury or amalgam was an easy operation. Old wooden launders from the tube-mill circuit were used for the fire and the ashes were smelted subsequently.

The subsequent treatment of the scalings may be determined according to their value. If rich, the method given on pp. 84 and 85 (*Ibid.*) will be found satisfactory, although it may be necessary to skim off slag and add more of the scaling-flux mixture in order to collect sufficient bullion to make a reasonable sized button or bar. If the scaling is

poor, as may be the case if the plates have been steamed and thoroughly scoured before scaling, then it is preferable to dissolve the copper in sulphuric acid before melting, and the copper may be recovered by precipitating on iron. Bullion containing over 850 parts gold per thousand should be obtained without difficulty.

What has been said about the clean-up of launders, etc., in the amalgamating plant largely applies to the cyanide plant from which accumulations of rich scale and fibre may possibly be obtained. Sand and slime should be separated from the fibre by drying and sieving or by washing, the sand being sent back for amalgamation and the fibre burned and smelted. This material may carry over 50 oz. gold per ton, and its collection and treatment should be a routine operation in an operating plant. The extractor house clean-up vats and appliances, together with smelting room and furnaces, all require close attention. Wooden extractor boxes may be burned and the ashes collected, those of steel may be chipped and moved aside so that the floor may be thoroughly cleaned. Dissolving and washing vats should be burned; filter presses, pipes, pumps, etc., thoroughly scraped and cleaned. Any small drains or sumps should be closely examined. All beams, rafters and walls in the smelting and fluxing rooms should be swept down. Retort and calcining trays should be scaled or chipped thoroughly; the furnaces and flues will have to be cleaned out, and the inside layers of bricks will in most cases carry some gold, so they should be collected and crushed.

The most payable method of obtaining value for the rich products collected in the extractor house and smelting room will depend upon local conditions. Most of the material can be smelted by the Tavener process if the furnaces necessary are available. A Tavener furnace itself will have to be carefully dismantled and the bricks sorted; a quantity of lead bullion is almost certain to have been absorbed by the hearth. It is generally advisable, however, to keep the necessary furnaces intact as long as possible in order to save the cost of shipping and smelting elsewhere. The smelting room floor will have to be pulled up and the ground surrounding the building should be examined, together with any old scrap, bricks, ash dumps, etc. Material unsuitable for smelting may be sluiced, ash dumps being re-burned if necessary.

The following approximate assay results taken after a plant had ceased operations, are of interest. It will be noted that the samples were taken mostly in the cyanide works since it was already known how valuable the deposits in the stamp-mill and tube-mill plants were:—

Sand from stamp-mill foundations: from 0.25 to 3.5 oz. gold per ton.

Scaling from tailing wheel elevator: 0.5 oz. gold per ton.

Scaling from sand and slime classifying cones: from 12 to 200 oz. gold per ton.

Scaling from sand and slime collecting and treatment vats: from 0.1 to 1.0 oz. gold per ton.

Scaling from sand and slime launders: from 10 to 20 oz. gold per ton.

Scaling from steel extractor boxes and screens: from 50 oz. gold per ton.

Scaling from extractor house floor: from 3 oz gold per ton.

Generally the clean-up of a plant can be done at comparatively little cost, particularly if barrelling, sluicing and concentrating methods are sufficient for the amount of material to be treated; all the appliances required are usually on hand or substitutes can be made from material no longer in use so that the cost of labour and water are the principal items, only a small amount of power being usually required, and for a time the operations may be highly profitable.

The experience and knowledge to be gained in carrying out these clean-up operations are valuable; it will be realised how easily amalgam and gold may be temporarily lost through insufficient care in an operating plant, which fact should be borne in mind by those in charge of reduction works operations, because opportunities for recovering some portion of that gold during the life of the mine do occasionally arise.

Mr. F. Wartenweiler: I take great pleasure in moving a vote of thanks to Mr. A. King for his timely and useful paper. Although the subject is not of a particularly scientific nature, it has considerable interest to many of our members.

Unfortunately a number of our mines have ceased operations during the past few years and the subject of the final clean-up of their reduction works, which embraces the recovery of all the gold left about on the surface, principally on the site of the works, has become an important one. One hears many extravagant tales of the riches of these cleans-up, but on close investigation it is

generally discovered that, providing care has been taken during the life of the reduction plant, the return is not fabulous. It would be interesting to us if Mr. King could supply data showing the amount of gold recovered in, say, the two definite stages of a clean-up, *i.e.*, the first stage, which includes the recovery of the absorbed gold in the extractor boxes and on the amalgamation plates; and the second stage, the recovery from the final or demolition clean-up, which includes all the launders, floors and the site generally.

It has been my experience that a five-stamp battery is useful in dealing with the large amount of material which contains payable gold, and a great deal of which requires disintegration. It may not be generally known that in the course of operations of this nature one handles a number of thousand tons of debris and soil. At one plant the entire mill yard for a depth of three feet was put through such a battery with a recovery of approximately half an ounce of gold per ton. Long blanket strakes have been found to be efficient collectors of the gold, and a concentrating table of the Wilfley type is useful in cleaning the concentrate from these blankets, producing an enriched product for sale to the Witwatersrand Co-operative Smelting Works.

Experiments have been conducted on the cyanide treatment of the tailings from the blankets. These were found to carry between 2 and 3 dwt. gold per ton, of which from 60 to 70 per cent. was dissolved in cyanide solution. As the units of the reduction works are usually too large for the cyaniding of this material, having in mind the comparatively small scale operations and as capital expenditure in erecting suitable treatment plant is generally not warranted, the gold in these tailings becomes a loss.

The recovery of gold from these cleans-up is also affected by the personal factor. The surroundings of the derelict or dismantled plants are of a lonesome character and have a depressing effect on the operator. Without disrespect to these men, all of whom are specially selected and are capable and conscientious, the experience on a particular plant was that the first, after about eight months' hard work, considered the job fairly well completed and was replaced by a second man who carried on profitably for another eight months, at the end of which time he turned over the practically completed work to a third party. The third party has been successful for four months, and expects to

continue for a much longer period. The work requires keenness and continuous following of enriched streaks or pockets, by panning. One has to realise the flour-like nature of much of the gold in the mill pulp.

The moral of these cleans-up certainly points to the necessity of system and organisation at those reduction works using amalgamation to prevent the dissemination of gold over the entire neighbourhood.

Mr. H. A. White said he would like briefly to second the vote of thanks to Mr. King for his able and excellent paper, and to state that it was at the special desire of the Council Mr. King had written on this subject, because they felt that details of operations of that nature, which had been going on in this district for some few years past, painfully frequently so, perhaps, should be placed on record. A great many people knew nothing about them. They felt their *Journals* would not be complete unless they had specially detailed accounts of operations of that nature, not only for future guidance when cleaning-up, but also to provide hints so as to avoid the chance of gold being left lying around in various places.

All of them who were metallurgists had had to listen to fairy tales of enormous quantities; in fact, great lumps of gold, and so on, being found about reduction works, but they would notice in Mr. King's paper and in Mr. Wartenweiler's comments thereon, that it was not in Bonanzas that the gold was found, but through careful search in every direction; although no great accumulations were found in special spots, yet quite a fair amount of gold was distributed over a considerable area of country; and, of course, the lesson they had to draw from that was, in future they had to be more careful—especially where the ancient and perhaps barbarous practice of amalgamation was still maintained. (Laughter.)

INVESTIGATION OF THE COPPER-NICKEL ORES OF THE RUSTENBURG DISTRICT.

By J. A. ORTLEPP.

(Printed in *Journal*, August, 1922.)

REPLY TO DISCUSSION.

The author wishes to thank Mr. Watson for his remarks on the paper, and as there

was no further discussion, there is nothing to be said in reply. However, a few notes on a portion of the paper may not be out of order.

The microscopic examination of opaque minerals is a subject which is deserving of some recognition. Its practical application is chiefly confined to the following two stages in the development of a mining enterprise, namely, prospecting and research in the case of complex or refractory ores.

To the prospector this method of determining minerals is a valuable addition to the stock of existing methods available, and as is the case with finely disseminated mixed sulphides, perhaps the only field method practicable. In the case of the copper-nickel ore examined the copper and nickel sulphides were invisible in the rough hand specimen on account of the close resemblance which their rough surfaces bore to the great mass of valueless pyrrhotite.

A knowledge of the relationship which exists between the various constituents of a complex ore may be of great assistance to an investigator who is attempting to obtain a suitable method of extraction of the valuable constituent. More especially is this evident where physical methods of separation are employed, and it is possible to foretell the behaviour of an ore to a certain extent when subjected to such treatment when the physical constitution is known.

SOME NOTES ON THE EVOLUTION OF THE KONIMETER.

By R. A. H. FLUGGE-DE SMIDT.

(Printed in *Journal*, November, 1922.)

REPLY TO DISCUSSION.

The first critic to whom I have to reply is Mr. C. J. Gray, and if all the criticisms had been conducted in the same spirit my task would have been an easy and a pleasant one.

Mr. Gray generously concedes several advantages of the circular type over the Kotze type, so I will confine myself to what he considers the disadvantages. The idea of the inaccuracy of the circular konimeter as compared with the original instrument

having once been mooted was probably fostered through making general comparisons between counts made by the Chamber of Mines officials as against those made by Government Mine Inspectors. The former used a 10 c.c. circular konimeter and counted with a light background, while the latter used a 5 c.c. Kotze konimeter and dark background illumination.

Both factors gave a higher count, in the latter case amounting very often to 100 per cent. with the combined factors. The accuracy of a konimeter is only relative, as Dr. Mavrogordato has pointed out, and I understood him to say that he got the same results whether using a circular or a Kotze konimeter.

In a letter to Mr. Boyd, dated December 14th, 1922, and circularised by the Chamber, Dr. Mavrogordato says:—

“As a result of comparing a considerable number of returns secured under carefully controlled conditions, Dr. Maynard came to the conclusion that variations associated with the spot were so great that one could disregard variations associated with the system of sampling. For instance, if samples were taken simultaneously with the Kotze pattern and the circular pattern there was no more difference between them than between two Kotze's or two circulars. There is no need to go into a long list of figures in support of these statements. In view of the absence of any supporting figures from Mr. Gray I must refute the statement of the comparative inaccuracy of the circular konimeter.

As regards leakage, I will refer to that point later.

As Mr. Gray in his camera simile has not proved the superior quality of the spot taken on the Kotze slide the one advantage of the plate disappears, and we are left with all the manifold advantages that roll-film possesses over the plate in handling, etc.

Besides, it is just as easy to take and observe a single spot on a circular disc as it is to take and examine fifty.

Mr. Gray's criticism of the Chamber's method in sending spots for counting to an independent observer has been ably answered by Mr. H. R. Adam.

I might add that the spots are then counted by an unbiassed expert.

Mr. Pirow states that changing slides in a dusty atmosphere underground did not appear to be a serious disadvantage with the Kotze konimeter. Mr. Pirow will no doubt

admit the risk of contamination, even if in practice it does not often happen.

Further on he states, when describing an improved form of spring clip: “The slide may be moved in any position under the small microscope without taking the slide out. That, of course, underground, where samples were taken under difficulties, appeared to be of great advantage.”

Of course it is of great advantage, hence the design of the circular konimeter to continue the motion without taking the slide out—and with a similar microscope attachment one can turn the disc backwards as well as forwards and refer back to any spot for comparison.

In reply to Dr. Moir's reference to the discrepancy between the figures of 30 per cent. of dust caught on the second disc and the percentages given after applying Dr. Owens' formula, I should have stated that I had applied the formula to *some* of the data collected by Mr. Boyd, for I had omitted a number of counts which were included by him when he obtained the figure of 30 per cent. caught on the second disc. The formula as such is a very simple one to apply.

In the February *Journal* I am quoted as saying: “There were various data giving the number of particles per c.c. which correspond to the milligramme per cubic inch” when cubic metre was intended. In the same connection perhaps I did not make it quite clear that in comparing counts with weight all counts referred to particles per cubic centimetre and all weights to milligrammes per cubic metre, so as to eliminate the million factor.

There is one mistake that has been pointed out to me, not during these discussions but privately, and that is where I refer to the experiments carried out by the United States Bureau of Mines.

I frankly confess I had not studied the article by Katz and Trostel in the *Journal of the American Society of Heating Engineers*, from which the data was culled, and my conclusions are entirely wrong. The sugar-tube caught 6.7 times, not 13 times, as much dust as the konimeter, and the ratio was fairly constant over high as well as low counts.

Leakages.—In referring to leakages Mr. Gray has said that even with the Kotze konimeter there was a danger of leakage, while Dr. Moir has stated that the Kotze instrument is the only konimeter which does not

leak in practical use. This latter statement cannot be substantiated.

In both types a glass slide makes contact with a rubber ring. Dr. Moir might just as well affirm that a large ship leaks but a small ship never. In practice I have found Kotze konimeters that leaked. During the last few months the rubber ring in the circular konimeter has been greatly improved, being rounded instead of flat, the increased compression attained thereby practically excludes all possibility of leakage.

Boyd's Experiment.—Nothing seems to have aroused so much heated discussion and even personal feeling as my innocent reference to Mr. Boyd's CaSO_4 experiment.

Mr. Boyd published the results of his experiment in the November-December Report of the Chamber of Mines Dust Sampling Committee. This I consider literature, and as such I am perfectly justified in quoting from it.

My remarks were confined to the micro-photographs which I had seen, while Mr. Boyd published the figures of the actual counts, namely—

1,000 + before acid treatment, and 650 after acid treatment.

First Mr. Gray, then Dr. Moir, and finally Mr. Pirow suggested possible errors in the carrying out of the experiment, and they were all tilting at windmills.

Here is a letter from Mr. Boyd, which will answer the criticisms and also explain why I was unable to produce the micro-photographs before the Society:—

Mr. Boyd writes: "I do not think any good purpose would be served by showing the micro-photographs of the calcium sulphate spot as they are the work of what is apparently only an amateur. But if any one cares to try the experiment let them make a solution of calcium sulphate in distilled water, spray some of it from a Tucker's Atomiser into the air, and take a konimeter spot in this spray. They will find a typical spot, which can then be treated with hydrochloric acid vapour as follows: Place a piece of filter paper, saturated with acid, in a small watch glass, invert this over the spot. It will be found that the spot does not disappear.

"In taking the micro-photograph the watch glass was over the spot all the time. I am not quite such an amateur as not to know how readily that part of the calcium sulphate which does go into solution would re-crystallize when the acid vapour was removed.

"It is an interesting and simple experiment, and can easily be tried by those interested."

As to the suggestion that Mr. Boyd should repeat his experiment before this Society, Mr. S. de Smidt might, with equal justification, ask me to get Dr. Owens to reproduce before this Society some of the tests to which reference had been made.

In amplification of his letter Mr. Boyd explained that on producing the micro-photographs someone would probably say they were out of focus. That he was fully justified in his surmise is proved by Mr. Pirow's gibe as to "spirit photographs."

Besides attempting to explain away the photographs, to be successful in discrediting the experiment as a whole one would have to explain away the counts by Mr. Boyd and Mr. McEwen.

Dr. Moir has given us a calculation wherein he computes that only half a milligramme of HCl is necessary to dissolve the maximum amount of CaSO_4 that can be present in one spot.

Let us remember we are referring to particles embedded in glycerine jelly and that according to Mr. Gray the solvent is absorbed by the jelly. Then let us visualise the quantities in terms of the microscope. Taking the thickness of a film of glycerine jelly as 10 microns, or perhaps 20 microns, and taking the area of a spot as one square millimetre (in reality it is less) then half a milligramme of HCl would represent a column of liquid over this area of nearly 500 microns in depth.

I appeal to the chemists present: Can this vast quantity of HCl attack the CaSO_4 without liquefying the jelly and causing it to run?

Dr. Mavrogordato's statement before this Society that if any adhesive had been used then *all* the CaSO_4 would not be dissolved has, I take it, finally disposed of this highly controversial matter.

Pirow's Final Contribution.—In Mr. Pirow's final contribution he gives some data that has a bearing on Mr. H. R. Adam's remarks regarding light and dark background.

Since Dr. Mavrogordato has dispelled the bogey of the ultra-microscopic particles and stated definitely that we are only concerned with particles down to half a micron in size then we should not count particles smaller

than half a micron, and the logical conclusion to be drawn from Mr. Pirow's data is to use light background only for counting.

Mr. Pirow then criticises my remarks regarding the tests carried out by the Chamber of Mines as being only possible through being able to use the circular konimeter.

His figures regarding the number of spots that he can take with a Kotze konimeter are very misleading. There is no limit whatever regarding the number of spots that it is possible to take with either type of konimeter. Mr. Pirow knows very well that you can multiply the number of spare discs that one takes underground with a circular konimeter, in the same way that one can take any number of Kotze slides. For routine work a single circular disc amply suffices.

As to the tests themselves, I really thought Mr. Pirow would have taken the larger view. The actual number of spots counted was only one factor.

Tests were not carried out on one mine, but on fifteen different mines. Each inspector had to send in his konimeter daily for an exchange of discs.

No mine inspector follows Mr. Pirow's practice of taking twelve spots on one slide. Three spots is a normal figure.

One Kotze slide to correspond with each gravimetric sample—and I maintain that it is sound practice—would have meant that every day 165 separate slides would have had to be prepared for treatment, each slide placed under a cover glass for half a hour's acid treatment, and then counted. Some spots taken at the delivery end of ventilation piping would never have been found on a Kotze slide.

As it was the microscope observers were taxed to the limit of their capacity. Using Kotze slides would have greatly multiplied their labours.

I do not doubt Mr. Pirow's ability to count four hundred spots in a day, but if he had to do that every day for two months he would end in a madhouse.

Adhesive on Slide.—All the references made by Messrs. Gray, Pirow and Atkin to the difficulty of applying a sufficiently thin film of adhesive only reinforce my arguments in favour of using a plain glass slide. Since I made that suggestion, Mr. Boyd has carried out a number of tests comparing spots on slides with an adhesive with those on plain glass. In all cases the counts on plain glass were higher than those on the sticky slides.

Dr. Mavrogordato has given us the history of his experiments with a plain glass slide, and has told us that the acid treatment is more effective when the spot is so taken. The spots themselves are beautifully clear and well defined, while there is less chance of contamination of the field.

Following up a tip from Dr. Mavrogordato, I carried out some experiments in which the spots were actually observed while under distilled water.

A fragment of cover-glass, moistened with distilled water, was placed over a spot immediately after acid treatment. By the time the focus was re-adjusted no movement of particles could be seen, although the count was greatly reduced. This I was compelled to attribute to change in visibility on account of the following observation. The room was warm and the film of water began to contract. After a time I was lucky enough to observe the receding line of water pass across the spot. Two points were interesting. Firstly the receding wave apparently did not move a single particle, and secondly the dry portion showed up doubly as dense in particles as the wet portion; in fact, the final count after the whole spot was uncovered was practically the same as before immersion.

On one occasion a number of tiny particles was left by the receding wave on the under surface of the cover-glass and, by adjusting the focus to them, an estimate could be made of the thickness of the film of water, namely 35 microns.

A point of interest is the extraordinary persistency of dust spots taken on a plain glass slide, and as Dr. Mavrogordato says, what is left of the spot after such treatment is assuredly insoluble mineral matter.

Another point is of great importance. On certain mines on the Far East Rand spots, whether taken by Kotze or circular konimeters, often show such a considerable contamination of the field that the spots are uncountable. By using a plain glass slide this contamination is very much less pronounced.

Conclusion.—Dr. Moir has accused me of being partisan. It was in all sincerity that I stated that if the Kotze instrument possessed any advantage over the circular konimeter, it was unknown to me.

I contend that in all the discussion that has resulted not a single advantage has been proved, and I adhere to my statement.

It is regrettable that so much personal feeling has entered this konimeter controversy. Each one of us is endeavouring to help towards the improvement of dust conditions, and I am sure that Sir Robert Kotze himself would be the last person in the world to decry any attempts at improving his original design simply because they do not emanate from himself.

RECOVERY OF GOLD BY BLANKET CONCENTRATION IN SUBSTITUTION OF PLATE AMALGAMATION.

By F. WARTENWEILER.

(*Printed in Journal, February, 1923.*)

DISCUSSION.

Mr. B. L. Gardiner (contributed): Mr. Wartenweiler is to be congratulated on a most interesting and comprehensive contribution, and speaking as one who has observed the methods of blanket concentration in a number of mills, it would appear from the description and figures submitted that the process has been brought to a greater degree of perfection at the Modder East than is usually the case where blankets or corduroy are employed. More particularly is this the case in respect to the smallness of the bulk that the gold is concentrated into before charging into the barrel, viz., 21.4 tons from 27,000 tons of ore, or .079 per cent.

Another feature is the four-hourly period between washing blankets—that is a long period compared with what is generally carried out, one-hourly or perhaps two-hourly intervals are more common—but the percentage of recovery as shown in the schedule per five blankets in series certainly proves that the long period is efficient at any rate for the purposes required at the Modder East. In the case of a treatment plant where blankets are the sole means of recovery it would no doubt be safer to wash more frequently in order to ensure maximum recovery, although at some little increase in cost.

In Rhodesia it is common practice to use cotton duck (filter cloth) in lieu of blankets or corduroy for the later series. Some millmen consider that fabric more efficient in

catching very fine gold, but apart from that it certainly retains less of the other minerals of the ore, and so is more desirable. Perhaps Mr. Wartenweiler would give his opinion on that point.

Note is made that the mercury consumption has been reduced by 66 per cent., and considering the greatly reduced tonnage of ore that is brought into contact with mercury by the blanket concentrating process, a greater saving might have been expected. Possibly the churning in the barrel with finely divided sulphides causes sickening and flouring of the mercury. In most places where blanket concentrating has been practised it has been a necessity owing to the usual plate amalgamation failing, either because the gold is coated or "rusty," or because other constituents of the ore react disadvantageously with mercury and the barreling of the concentrate, often with chemical reagents, is required to bring about the union of gold and mercury. In the case of a Rand ore there would appear to be no such disability, and it would be interesting to know whether the gold in the concentrate would not amalgamate by passing over a plate perhaps several times and whether the mercury consumption would not be less by such method.

Mr. E. M. Weston (contributed): Mr. F. Wartenweiler has presented a most valuable record of achievement in improvement in metallurgical practice on this field, which, however long delayed, yet marks a distinct advance on previous practice.

Mr. Wartenweiler's review of past practice and of past literature in connection with this subject is somewhat scanty. It is a pity that some Rand metallurgist several decades ago had not visited the St. John del Rey Mine, in Brazil, or other localities mentioned by me in various papers and contributions to discussions on this subject before the Society. I had the opportunity recently to experiment on the lines suggested by me in my paper "Can Amalgamation be dispensed with?" on a small mine crushing ore in which some of the gold was in a very fine state of division. The lesson learnt may be of value to small workers who contemplate trying this method. I expressed myself in favour of using some device to take out the bulk of the coarse gold at once, and thus relieve the burden on the blanket tables. I suggested and sketched several devices, but I adopted finally in practice a very cheap, simple and effective one. I took three boards:

of clean pine, about one and one-eighth inch thick and about twelve inches in width, of the same length as the width of the table. These I ruled with parallel lines an inch apart and at two-inch centres, staggered on alternate rows; with an auger I bored holes about a quarter to three-eighths of an inch deep and burned them out to remove roughness. In the centre of each hole I drove a short stout nail, so that the head stood out about one-eighth of an inch above the level of the board. Attached to the sides of the mill-table by hinges on one side and hasps and staples and padlocks on the other, were three corresponding pieces of plank which served as locked covers and prevented any attempt at theft. Below these three concentration boards were the blankets, which may be tacked to frames for convenience in changing and washing, and which are thus always kept stretched. I found that the boards retained all the coarse gold and much of the very fine gold, and could be left for twenty-four hours, while in normal circumstances the fine gold took eight hours to work down to the end of the blankets. The drawback, as far as small mills are concerned, is this: These mills are usually run mostly by kafirs, especially on night shift. If the mill water is turned into the box, after a stoppage of the mill for any reason, before the stamps are dropped, a heavy wave of water is dashed down the blankets when the stamps are dropped, and this has the effect of washing fine gold over the blankets. If this method is adopted both kafirs and white attendants must be trained to on no account allow any such rushes of water to occur. Mr. Wartenweiler notes that blankets are more efficient than canvas tables for concentrating battery sand. This has, of course long been recognised in ordinary concentrating practice, but canvas tables are much more effective than blankets for concentrating classified slimes, and in some cases slime treatment might be avoided by using a large area of easily manipulated canvas tables on the Australian system. As I have before related, I have seen gold so fine that it would float for several minutes in water caught on such tables. Often in small works the amount of slime produced does not warrant the instillation of a slimes plant, and in such a case an installation of canvas tables saves slime gold and slime sulphides.

NOTES ON CUPELLATION LOSSES.

By H. R. S. WILKES.

(*Printed in Journal, February, 1923.*)

DISCUSSION.

Mr. R. A. Cooner: In connection with this paper the presence of a dark stain on the surface of the cupel immediately underneath the silver-gold bead has been noted. I should like to say that I have analysed material scraped from the stained surface and found it to contain a very appreciable amount of nickel. It is well known now that some parts of the Reef contain considerable quantities of nickel and cobalt and apparently traces of these metals are frequently reduced during assaying and resist the oxidising action of cupellation to the last.

The stain, in addition to nickel and possibly cobalt, frequently contains also a quantity of silver which may be carried down mechanically by the base metals. Minute particles of silica in the cupel surface appear to attract nickel—possibly by offering an acid to flux the nickel, and I think that this accounts for occasional greenish stains on the higher surface of the cupel. If these stains are examined under the microscope there will usually be found a minute bead of bullion in the centre. In fact, I think that assayers would often be startled to see a cupel surface under the microscope, especially when very rich samples or bullions have been done. Although the bullion assay may be entirely satisfactory, and no abnormality exist in the weights of the check sample of pure gold, the microscope will usually reveal an amazing number of minute beads of silver or gold in the pores of the cupel surface. The accuracy of the determination, which is undoubted, may be due to a fortunate balance of errors in the method which is universally employed.

THE SOLUBILITY OF GOLD AMALGAM IN CYANIDE SOLUTIONS.

By H. A. WHITE.

(*Printed in Journal, March, 1923.*)

DISCUSSION.

Mr. B. L. Gardiner (contributed): Mr. H. A. White's very interesting paper on the above subject certainly gives rise to

much speculation, particularly in regard to his suggestion as to the solution of the problem of the lower residues in the absence of mercury. In the preliminary experiment it would appear that the dissolving rates, viz., 9.5 mgm. in the case of gold, and 9.4 mgm. of amalgam in the case of amalgam, were much about the same, and the increase in time required to dissolve a given piece of gold coated with mercury would be proportional, approximately, to its increase in bulk.

In the major test the pure gold loses 166.6 mgm. while the amalgamated strip loses only 77.5 mgm. of gold and 7.5 mgm. of mercury. The lower dissolution rate of the mercury is understandable as there is a rapidly diminishing quantity of that metal present, at the end of the test only 3.7 mgm. remaining ($16.3 - 5.1 - 7.5 = 3.7$).

For some reason, however, the gold, despite the removal of the mercury, appears to be less readily soluble relatively; a fact difficult to understand. However, accepting the relative solubility rates of clean and amalgamated gold as being as 2:1 there is much greater difficulty in accepting the hypothesis that the gold particles in the tailing are coated to any appreciable extent with mercury. The tentative suggestion that mercury may behave similarly to oils used in froth flotation is most difficult of acceptance; the tendency for oils to emulsify and form milky mixtures with water is not imitated by mercury. Evidence is accumulating that the use of mercury *does* affect the final residues, and it is possible that the coating of some of the particles of gold by that metal is a factor, but there are probably other factors perhaps of greater importance. It is well known that in all mills a certain amount of mercury is carried away in the tailing, and millmen will admit that on panning average samples of tailing they rarely see mercury, certainly never anything approaching 0.1 oz. per ton. It is obvious that the mercury must therefore be either in an exceedingly fine state of subdivision, or else is attached to some particles of ore.

Mercury when added to a plate is rubbed in and immediately becomes mixed with a certain amount of gold; the stream of ore and water passes over, and while nearly all of the freed gold particles adhere and are absorbed, a certain amount of amalgam is detached and carried away. It is well known that certain ores, notably those carrying appreciable quantities of pyrrhotite and mispickel carry off large quantities of mercury

(or amalgam) from plates, and in these extreme cases the following is what happens. The bright plate rapidly becomes darkened, a brown or black film forming on the surface, which is not very cohesive with the plate and is readily removed by the scour of the passing ore particles. On one occasion when conducting some experimental work the writer recalls that the amalgam was completely stripped off the plate—the tailing gave a dark-brown tail on panning that resembled ground coffee in appearance, which, on close examination under a magnifying glass, proved to be minute globules of mercury each covered with a film, presumably of a base metal sulphide. On cyaniding this tailing extremely high residues resulted, whereas ore treated by first passing over blankets and then cyanided gave good residues.

All base metal sulphides amalgamate with mercury to a greater or lesser extent, and it might be surmised that some such action as just described may take place with most ores although in a much less marked fashion, and not so readily recognisable. If it does, then it is not merely a question of minute globules of mercury carrying gold being carried into the tailing (such small globules would not offer any great difficulties to solution cyanide even if the solution rate is half that of gold) but if the globules are coated with something not readily soluble in cyanide then a very different state of affairs exists and gold contained in such globules may fail to become dissolved.

These suggestions are put forward as an alternate hypothesis as to why disuse of mercury should be the means of decreasing the value of residues.

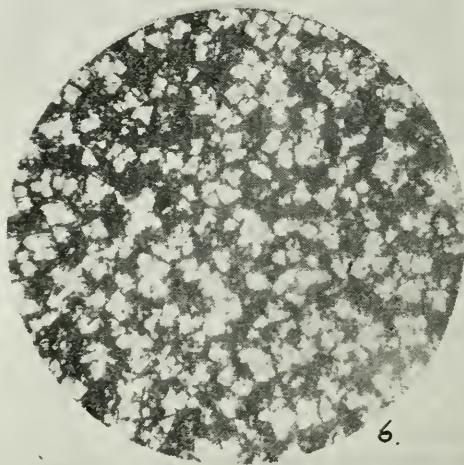
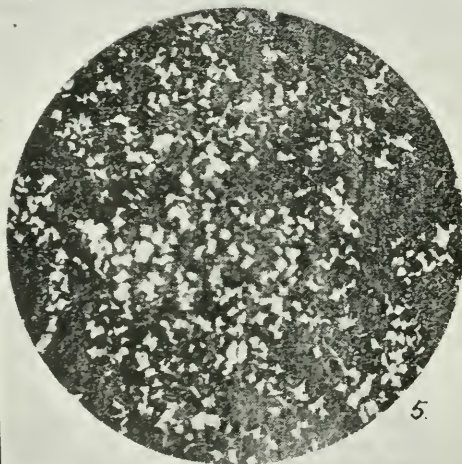
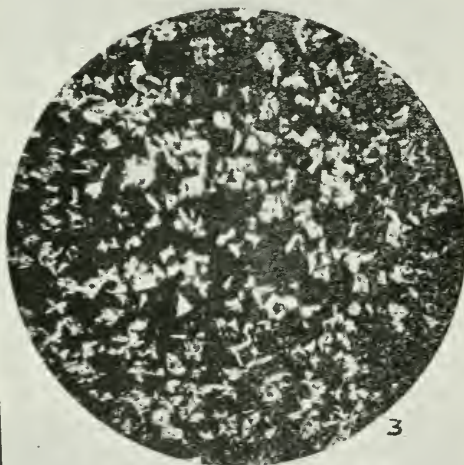
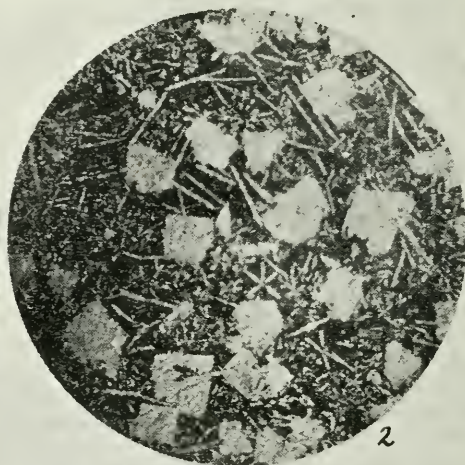
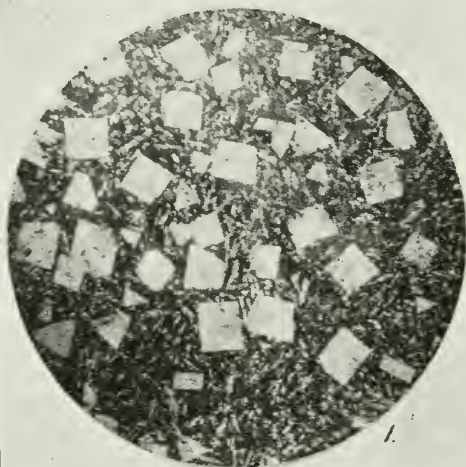
OBSERVATIONS ON THE TIN-ANTIMONY-LEAD ALLOYS.

By R. W. IRWIN.

(Printed in *Journal*, March, 1923.)

DISCUSSION.

Mr. R. A. Cooper: I should like to add a few further remarks on this subject in view of Mr. Sharpe's contribution to the discussion. I have been fortunate in having



had the opportunity of discussing these alloys and their performance with Mr. Sharpe during the three year test, and my views are well known to him, but I should like to endeavour to stimulate others to express an opinion also.

The alloys mentioned by Mr. Sharpe were:—

Alloy No.	Tin.	Lead.	Antim.	Copper.	Zinc.
1	83	—	11	6	—
2	60	22	12	6	—
3	5	79	14	—	2
4	—	80	20	—	—
5	5	80	15	—	—

It is not possible to calculate the exact percentage of the metallic compounds, etc., which are present in each alloy, as equilibrium is not attained under ordinary foundry methods of preparation.

I think, however, that the following very closely approximates to the constitution of the alloys as usually prepared:—

Alloy No. 1.—Copper: Tin crystals (in skeleton form), 9 per cent.; Antimony: Tin crystals (hard cubes), 15-17 per cent.; Tin containing a little antimony (ground mass), 75 per cent.

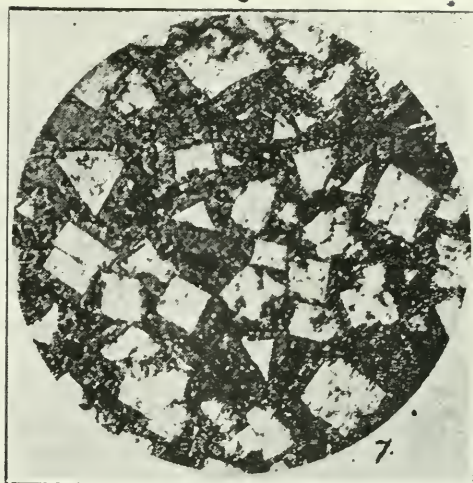
Alloy No. 2.—Copper: Tin crystals, 9 per cent.; Antimony: Tin crystals, 20 per cent.; Lead: Tin eutectic (ground mass), 71 per cent.

Alloy No. 3.—This alloy and No. 5 are so nearly identical in composition that it is difficult to account for their difference in practice. The zinc would probably form a zinc-antimony crystal similar to the antimony-tin. This may possibly be an example of the influence of small amounts of one element on the properties of others. Zinc generally increases the tendency of a bearing metal to "seize."

Alloy No. 4.—This is a mass of antimony-lead eutectic with crystals of almost pure antimony distributed through it.

Alloy No. 5.—The addition of tin to this alloy produces an antimony-tin crystal such as is present in No. 1 and No. 2, instead of the pure antimony crystal of No. 4.

This antimony-tin crystal has been found to be the best "hard-constituent" of white bearing metals. It is found in all these alloys except No. 4, in proportions varying roughly between 10 and 20 per cent. of the whole.



The antimony-tin crystals form about 10 per cent. of No. 5; the ground mass being antimony-lead eutectic, and therein it differs from the tin of No. 1 and the lead-tin eutectic of No. 2.

It is interesting to note that the lead-antimony eutectic is giving at least as good results as the tin, and better results than the lead-tin eutectic as a base.

The micro-photographs show the structures of Alloys Nos. 1, 2, 4 and 5, and one other.

In the first two the copper-tin skeleton crystals, which act as a stiffening, can be seen between the antimony-tin cubes. The third shows the antimony-lead alloy No. 4 as normally cast at a low temperature, and the fourth gives some indication of the change effected by casting the same alloy from too high a temperature.

The fifth photograph is of Alloy No. 5

It will be noticed that Alloys No. 1 and 5 correspond closely to the Chamber of Mines standard metals Nos. 1 and 6. The last two photographs are a further illustration of the effect of casting temperature on the structure of bearing metals. These are of the Chamber of Mines Alloy No. 4 cast at the correct and at too high a temperature; in each case the mould was heated to approximately 100 degrees C. before casting.

The photographs are all taken at the same magnification ($\times 100$ diam.), with the exception of the first two, which are magnified to a greater extent to show the structure more clearly.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

ELECTRODEPOSITION OF IRON.—Eight different types of deposits are described in a fifty-page Bulletin which has been written for the British Department of Scientific and Industrial Research, namely, normal, columnar, mosaic, poplar-tree, fibrous, fan, cauliflower and fountain. Most of these names are self-descriptive and applicable to various electrodeposited metals. Normal deposit is most frequent; in it many prominent grains have a V-shape, pointed inward toward the cathode. Using a current density of 120 amp. per sq. ft., a $2\frac{1}{2}$ gallon bath of neutral ferrous chloride,* depositing for two hours on a cleaned copper rod, the effect of varying temperature from 70 to 116 deg. C. is as follows: As temperature increases the surface becomes rougher, and the grain becomes coarser, the deposit softer and less brittle. (It is a quite general rule that the more coarsely crystalline the surface is the larger the grain size of the metallic deposit.) Maintaining the temperature at 110 deg. C., and varying the current density from 60 to 240 amp. per sq. ft., gave results indicating that a maximum grain size is reached at about 120 amp. per sq. ft., and a minimum exists at about 200. Heavier densities again coarsen the grain. As before, coarse grain is associated with softness and lack of brittleness; the metal can be safely hammered while cold. A great and striking difference exists between two deposits made under like conditions except that one contains a rotating cathode. The latter surface is extremely smooth, and the metal is very much finer in grain, often approaching a fibrous structure when sectioned and examined under the microscope. It is also characteristically hard and brittle. Adhesion of the deposit depends upon absolute cleanliness of the cathode. This involves boiling in an alkaline cleanser, treatment by an electro-cleaner, and dipping in 10 per cent. HCl, with appropriate scrubbing and rinsing between. A small quantity of free acid will generally cause an iron deposit to be fine grained to fibrous, even if made at low current density. Furthermore, the solutions must not become materially depleted in iron content if thick, uniform deposits are desired.—W. E. HUGHES, *Chem. and Met. Engineering*, Vol. 27, No. 14, October 4, 1922, page 705. (J.A.W.)

THE PROPERTIES OF PURE HYDROGEN PEROXIDE.—Starting with commercial 3% hydrogen peroxide solutions, containing many impurities, it has been shown how the pure hydrogen peroxide can be isolated. The methods present several novel features, and a 50% yield of pure hydrogen peroxide is possible.

The freezing point was found to be $-1.70^{\circ}\text{C}.$; the density of the anhydrous liquid 1.4633 at 0° , and 1.4442 at 18° , and of the solid 1.6434 at -4.45° ; the viscosity in absolute units 0.01828 at 0° .

* Concentration not noted; possibly Merck's solution (100 kg. FeCl_2 to 100 litres of water).

A number of other physical constants were determined, and miscellaneous facts about the anhydrous peroxide stated. One molecular compound with water, namely, $\text{H}_2\text{O}_2 \cdot 2\text{H}_2\text{O}$, with a melting point of $-51^{\circ}\text{C}.$, has been shown to exist.

The effects of pure peroxide on the skin were observed and it was found that living tissue was not hurt in the slightest but dead tissue was destroyed and removed.—Part I., O. MAASS and W. H. HATCHER, Part II., O. MAASS and O. W. HERZBERG, *McGill University Publication*, Series 111., No. 22. (R.A.C.)

ESTIMATION OF SULPHUR IN IRON AND STEEL.—A weighed quantity (0.25 gm.) of fine drillings or borings is placed in a minute glass tube closed at one end. This is introduced into a small flask provided with a ground-in glass tube suitably bent, and dipping to the bottom of a 70 c.c. cylinder half full of 2% Na HO solution.

Seven c.c. of hydrobromic acid are added to the flask, the glass tube connected and the sample brought into contact with the acid, and gradually heated; boiling is continued till solution is complete.

The cylinder is then lowered and the tube rinsed down. The liquid is treated with 10 c.c. of acetic acid (250 c.c. of glacial acid per litre) and starch solution, and the sodium sulphide immediately titrated with 0.066 N iodine solution. Not more than 4 c.c. are required in the titration.

The method is claimed to be accurate, and very quick (20 to 25 minutes), the results are higher than those obtained by the usual method. This is explained by the more complete conversion of the sulphur into hydrogen sulphide by the hydrobromic acid.—F. NIKOLI, *The Analyst*, (abstracted from *Chem. Zeit.*, 1922, 46, 1025-1026, January 1923), Vol. XLVII., No. 562, p. 43. (H.D.B.)

TARS FOR ROAD MAKING.—The Ministry of Transport (Roads Department) have recently recast the official publication entitled "General Directions and Specifications Relating to the Tar Treatment of Roads." This may be purchased from His Majesty's Stationery Office, price 1s. net. The revised Schedule of Requirements and Methods of Testing are detailed in the *Chemical Trade Journal*. The latter portion deals with official methods of determining specific gravity, phenols, naphthalene, free carbon, etc., and describes the required distillation test.—MINISTRY OF TRANSPORT, *Chemical Trade Journal*, February 16, 1922, p. 191. (R.A.C.)

BOOK REVIEW.—THE ANALYSIS OF NON-FERROUS ALLOYS. By Fred Ibbotson and Leslie Aitchison. London: Longmans, Green & Co., 39, Paternoster Row, E.C.4. (Price, 12s. 6d. net.)—The first edition of this book was published at the beginning of the war, which did a great deal to encourage the control of metals by chemical analysis. Besides this, the war enhanced the importance of many alloys which previously had been practically unknown or very little used, such as the extended use of non-ferrous alloys, however, there was not any concomitant development in

the methods used in their analysis. The aim of the present second edition of this book is to provide the best analytical methods of such new alloys brought into prominence since 1914. Among the new materials added are methods for the analysis of the more important alloys of aluminium, nickel, zinc, etc. Other notable additions deal with the determination of special elements, such as iron and manganese that are added to bronze in order to give it mechanical properties of special value.—*Iron and Coal Trades Review*, October 13, 1922. (J.A.W.)

METALLURGY.

THE EMPIRE'S RESOURCES IN THE RARER METALS.—Abstract of Presidential Address of Professor G. T. Morgan, delivered to the Chemical Society of the University of Birmingham.

British sources of tungsten, barium, strontium, thorium and cerium are sufficiently large to supply our own demands and also to supply a considerable proportion of the world's requirements.

Zirconium has been examined chiefly by German and American chemists, and is worthy of further research. The value of zirconia as a refractory has been clearly demonstrated, and many other uses have been indicated. There is a world famine in regard to the valuable metals of the platinum group owing to the failure of the Russian supply. Tasmania has furnished osmium and iridium, and the Sudbury ores of Ontario yield platinum and palladium.

About four hundred tons of cobalt are produced annually, and the demand is likely to increase.

The market for magnesium is dominated by Germany, which controls the Austrian magnesite, and is able to undersell any competitors.

The author discusses the metallurgy of beryllium at some length. Much research on this metal and its compounds remains to be done. Even a moderate supply of beryllium or its alloys with copper and other metals would be an aid in aeroplane and airship construction. Deposits of its commonest mineral compound, beryl, occur at Limoges in France, in New Hampshire, in County Dublin and in South-West Africa.

Of the light metal group only aluminium is plentiful; some of the others, such as scandium, gallium, indium and thallium are excessively rare.

Yttrium and lanthanum belong to the metals of the rare earths, comprising some eighteen elements, of which thorium and cerium have met with industrial application.

Titanium is not yet produced commercially as an element, but is used in the production of ferro-titanium alloys. The oxide titania is used in pigments and in glass.—PROFESSOR G. T. MORGAN, *Chemical Trade Journal*, January 26, 1923, p. 100. (R.A.C.)

BOOK REVIEW.—AN ELEMENTARY TEXT-BOOK OF METALLURGY. By A. Humbolt Sexton. Sixth edition; revised by C. O.

Bannister. London: Charles Griffin & Co., Ltd., Exeter Street, Strand, W.C.2. (Price, 8s. 6d. net.)—The resignation of the original author of this popular text-book from the professorship in Glasgow led to Professor Bannister undertaking the work of revision and bringing the text up to date. It has been found necessary to make considerable corrections in connection with the properties of metals; also to introduce a classification of pyrometers and new matter on the manufacture of coke and producer gas. The subject of drying the air for blast furnaces and the cleaning of blast furnace gas has received attention; whilst the latest information as to the treatment of steel ingots and of special steels has been added. The chapter on the application of electricity to metallurgy has been almost entirely re-written. The laboratory course, Part II., however, comprising practical exercises, has been left almost intact with only minor alterations. Special attention is given throughout the book to principles; select processes only are therefore described, and no more details given than are necessary for the understanding of the principles.—*Iron and Coal Trades Review*, October 13, 1922. (J.A.W.)

MINING.

BOOK REVIEW.—THE MINING LAWS OF THE BRITISH EMPIRE AND OF FOREIGN COUNTRIES, Vol. III. South Africa. Part I. General Principles applicable to South Africa: The Transvaal. By Gilbert Stone. London: Stationery Office, Imperial House, Kingsway, W.C.2. (Price, 30s. net.)—This is one of the volumes prepared for the Governors of the Imperial Mineral Resources Bureau, on the subject indicated in the title, the intended publication of which volumes was announced in these columns some years ago. The other volumes in the series already published are: Vol. I., Nigeria; Vol. II., West Africa; and Vol. IV., British Columbia.—*Iron and Coal Trades Review*, December 22, 1922, p. 932. (J.A.W.)

COMPRESSED AIR OPERATED VENTILATION DOORS.—E. D. Gardner (mining engineer, U.S. Bureau of Mines) reports that a number of metal mining companies in Arizona are using ventilation doors which can be opened and shut mechanically by the motorman without leaving his seat or stopping his train, and thus interfere very little with the transportation of the ore.

The doors of the Calumet and Arizona Mine in the Bisbee district are representative of those in use. The doors at this mine are substantially built of 2in. lumber, are airtight, are well placed, and have two openings or passageways. The main door, which is used for the motor trains, is 5ft. wide by 6ft. high, and is opened by compressed air and closed by a counterweight. At the side of the motor door is the man door, which is 18in. wide by 5ft. high, and is opened and closed by hand.

The mechanism for opening the main door consists of a section of 5in. pipe about 5ft. 6in. long, threaded at both ends and used as an air

cylinder. A piston fits in the cylinder and is attached to a 1-in. plunger rod, which passes through a stuffing box at one end of the pipe. The free end of the rod is attached to a rope that goes over a pulley and is fastened to the door. A $\frac{3}{4}$ -in. pipe connection from the compressed air line is tapped into the cylinder at the packing end. When the air is turned on, it forces the piston to the other end of the cylinder and thereby opens the door. The door is held open as long as the air pressure is maintained, and is closed by the counterweight and the force of the air current when the pressure is released. The air is turned into the cylinder from the main air line through a three-way cock. Levers placed at some distance from and on either side of the door are connected to the air cock by two wires. When a train approaches the door, the motorman reaches out and pulls the first lever; this opens the cock, turns the air into the cylinder, and opens the door. After the train has passed through the door, the motorman pulls the lever on the other side, which shuts off the air, releases the pressure in the cylinder and the door is closed. The motorman on a train coming from the opposite direction would pull the same levers, but in reverse order, as these are so arranged that the doors are opened or closed by pulling the levers in the direction of the moving train. The distance that the levers are placed from the door depends upon the length and speed of the trains, but should be long enough to give the motorman time to stop his train before it wrecks the door if the mechanism failed to work. At the Calumet and Arizona Mine the distance is 150 ft.

A red light is generally placed at each lever and other safety precautions taken. Some doors have been demolished in some of the mines by trains striking them, but with ordinary care on the part of the motorman the use of the doors is safe and no accidents should occur.

The Magma Copper Co., at Superior, Arizona, uses a modification of the same door in the main haulage level. As no supplemental man-doors are used, the danger of a man being struck when the doors are opened was obvious, and whistles, which start to blow as soon as the air pressure is turned on, were placed in the air cylinders, and give warning to all in the vicinity. The whistles also continue to blow as long as the doors are open, and remind all in that part of the mine to release the air. E. D. GARDNER, *Colliery Guardian*, June 9, 1922, p. 1424. (J.C.)

EXTRACTION OF COAL AROUND SHAFT PILLARS. G. Marbach (*Gluckauf*) states that some 60 out of the 564 shafts in the Ruhr coal basin have suffered injury—mainly deviations from the vertical—due to collapse of the shaft supports, and that costly repairs have from time to time become necessary. The forces acting to disturb a shaft subject to various conditions are shown in diagrams, and illustrated by actual occurrences of shafts thrown out of the vertical in flat, in moderately inclined, and in highly inclined strata, and with walled and timbered pit eyes. It is shown that unsupported rock beds, if of shaly materials, give way slowly and by bending; while sandstone beds remain

apparently intact until they fracture suddenly. Thus the shale indicates the existence of stresses before they have developed in a dangerous degree. Excavations in the immediate neighbourhood of the shaft should not be left long unsupported. The sinking of the strata in contact with the shaft casing may cause a destructive amount of friction. The more even and smooth therefore the outer face of the lining is, the less that friction will be. From the point of view of stability, brick walling is much more effective, as well as more durable, than a timbered lining. The lack of flexibility in a brick casing may be remedied to a large extent by building the casing in sections of, say, 100 to 150 ft., separated by a ring of soft wood about 12 in. thick. From this point of view, reinforced concrete for shaft lining has not realised all the hopes formed concerning it. All things considered, cast iron or steel tubing appears to fulfil more completely the conditions which have to be met in shaft construction. The author refers to methods for giving some degree of flexibility, such as wedging with hard wood and, in certain circumstances, forming alternate rings of iron tubing and brick walling. These questions, together with the form and dimensions of the shaft pillars, are discussed exhaustively.—*Colliery Guardian*, June 9, 1922. (J.C.)

CARBON MONOXIDE ASPHYXIA.—Experiments carried out in America on the treatment of carbon monoxide asphyxia have shown that when breathing oxygen the rate of elimination of carbon monoxide is somewhat more rapid than when the subject breathes merely air, but that the elimination is enormously accelerated when the inhalation consists of oxygen, plus carbon dioxide. A blood concentration of carbon monoxide of 40 to 50 per cent. saturation (a dangerous amount) was reduced to 10 or 12 per cent. (a quite harmless amount) in half an hour.—*Indian Engineering*, December 16, 1922. (J.C.)

CHANGES OF ADDRESS.

- HEBBARD, A., *U/o* Benoni: P.O. Box 12, Knights, Transvaal.
- LINDEMANN, W. C., *U/o* Rissik Street, Johannesburg: 38, Bedaerts Street, Troyeville, Johannesburg.
- MEYER, T. C. A., *U/o* Modderfontein Deep Levels, Ltd.: Geduld Proprietary Mines, Ltd., P.O. Dersley, Transvaal.
- REDFERN, A., *U/o* Grenfell Colliery, Ermelo: Townlands Colliery, P.O. Box 125, Ermelo, Transvaal.
- ROBINSON, V. E., *U/o* Globe & Phoenix G. M. Co. Ltd. Que Que, S. Rhodesia: Rothamstead Mine, P.O. Box 34, Que Que, S. Rhodesia.
- SMITH, G. Hildick, *U/o* Robinson G. M. Co. Ltd., Crown Mines, Ltd., P.O. Box 102, Crown Mines, Johannesburg.
- WARD, H., *U/o* Roodepoort United Main Reef G. M. Co. Ltd., Aurora West United G. M. Co. Ltd., P.O. Box 26, Maraisburg, Transvaal.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA

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Vol. XXIII.

JUNE, 1923.

No. 12.

Proceedings

AT

**27th Annual General Meeting,
23rd June, 1923.**

The Twenty-Seventh Annual General Meeting of the Society was held in the Assembly Hall, Scientific and Technical Club, 100, Fox Street, Johannesburg, on Saturday, 23rd June, 1923, at 8 p.m., Prof. G. A. Watermeyer (President) in the chair. There were also present:—

21 Members: Messrs. C. J. Gray, F. W. Watson, J. R. Thurlow, J. Chilton, R. A. Cooper, A. King, Dr. A. J. Orenstein, J. M. Thorburn, John Watson, H. A. White, J. A. Woodburn (Members of Council), H. D. Bell, J. A. Boyd, H. T. Brock, J. M. Dixon, J. Henderson, G. Melvill, P. T. Morrisby, J. M. Muller, T. K. Prentice, R. A. H. Flugge-de Smidt.

6 Associates: Messrs. C. E. Deakin, O. A. Gerber, R. W. Irwin, R. J. B. Kethel, H. Rusden and W. Russell.

4 Visitors and H. A. G. Jeffreys (Secretary).

MINUTES.

The Minutes of the Ordinary General Meetings held on 21st April and 19th May, 1923, as reported in the respective *Journals* of those dates, were confirmed.

NEW MEMBERS.

A ballot was taken for the election of new members, and the following gentlemen were declared duly elected:—

COLLINS, HARRY, New Primrose G.M. Co., Ltd., P.O. Box 193, Germiston, Battery Manager.
GRIER, CHARLES D., c/o Messrs. Fraser & Chalmers (S.A.), Ltd., P.O. Box 619, Johannesburg, Metallurgist.
MCCALL, GEOFFREY, Rand Club, Johannesburg, Engineer.
QUINAN, KENNETH B., P.O. Dynamite Factory, Somerset West, C.P., Chemical Engineer.
(Transfer from Associate Roll.)

The Secretary announced that the Council had admitted as Associates the following gentlemen:—

COIBORNE, G. T. B., New Modderfontein G.M. Co., Ltd., P.O. Van Ryn, Transvaal, Mining Engineer.

McKUNE, L. M., P.O. Box 7017, Johannesburg, Acting Manager, Baldwin Locomotive Works, Johannesburg.

The Secretary read the Annual Report, as follows:—

ANNUAL REPORT OF COUNCIL, YEAR ENDING
JUNE, 1923.

Your Council has great pleasure in submitting the Annual Report upon the work of the Society during the past year, together with the Balance Sheet and Statement of Revenue and Expenditure. This is the Twenty-seventh Annual Meeting of the Society and the Twenty-ninth year of its history, the South African War having prevented Annual Meetings being held in 1900 and 1901.

Accounts.—From the accounts before you it will be seen that the result of the year's working shows a surplus of Revenue over Expenditure of £108 7s. 11d. This compares satisfactorily with last year's profit of £84 1s. 10d. and the losses shown for the two previous years. On the expenditure side of the account general charges are higher than last year, while printing and stationery are correspondingly less; the salaries item is £20 lower. The total working expenses have been reduced by £22 5s. 5d. Your Council has granted the usual donation of £25 to the Seymour Library and has also contributed to the Vice-Chancellor's Medal Fund for the Witwatersrand University. In the way of exceptional expenditure the sum of £34 11s. 5d. represents the landed cost of the War Memorial and £33 13s. the balance of the Assessment by the Associated

	June, 1899	June, 1902	June, 1903	June, 1904	June, 1905	June, 1906	June, 1907	June, 1908	June, 1909	June, 1910	June, 1911	June, 1912	June, 1913	June, 1914	June, 1915	June, 1916	June, 1917	June, 1918	June, 1919	June, 1920	June, 1921	June, 1922	June, 1923
Hon. Members	14	9	13	18	18	16	17	18	21	22	20	23	25	25	26	26	25	23	22	21	24	24	25
Life Members	1	1	1	2	3	4	7	7	8	9	9	9	10	10	10	9	9	9	9	10	10	10	11
Members	281	291	441	639	680	680	683	618	630	731	711	733	618	550	481	463	454	469	463	340	363	370	360
Associates	238	243	281	338	321	314	342	357	342	402	380	442	396	351	287	272	263	270	270	178	166	165	131
Students	—	—	—	4	8	4	7	18	20	60	57	22	21	15	2	3	5	2	2	6	—	1	2
	534	544	736	1,001	1,030	1,018	1,056	1,018	1,021	1,224	1,177	1,229	1,070	951	806	773	756	773	766	555	563	570	529

Scientific and Technical Societies of South Africa not yet recovered from members. As regards revenue, members' current subscriptions are practically the same, while those recovered from associates are £22 less than last year. Arrear subscriptions have increased, and the aggregate of subscriptions collected shows an improvement to the extent of £15 12s. 9d. Entrance Fees have increased, while the profit on the *Journal* is £216 4s. 3d., as compared with £161 0s. 9d. Transactions account shows a profit of £14 9s. 3d., as against the small loss shown last year.

Turning to the Balance Sheet, it will be observed that while our cash resources are some £50 higher, liabilities to creditors are lower by over £120. The amount due by sundry debtors is £45 lower.

Membership.—Forty-seven new members have been elected during the year, of whom thirteen have been transferred from the Associate Roll, and your Council has admitted thirteen Associates and two Students. Thirteen members and seven associates have resigned, while your Council has to record with deep regret the deaths of Messrs. H. S. Meyer (Past President), H. S. Kenney and J. Wood. As a mark of esteem and appreciation of the many services rendered by him to the Society your Council has elected Dr. James Moir an Honorary Life Member.

The names of certain members and associates whose subscriptions have fallen more than two years in arrear have been struck off the roll; your Council has taken this step with reluctance, but the fact that the Assessment by the Associated Scientific and Technical Societies is calculated on the membership roll has rendered it imperative that strong action be taken in order that the Society shall not suffer pecuniary loss through its defaulting members.

Papers and Journals.—There has been a gratifying increase in the number of original papers read before the Society. From the list given hereunder it will be seen that, apart from contributions to the Miners' Phthisis Symposium, twenty-two papers have been published, as compared with fourteen during the previous year.

The following original papers have been published during the year:—

July, 1922:—

"The Application of Flotation to the Antimonial Gold Ores of the Murchison Range," by H. R. Adam.

August, 1922:—

"Statistics of Extremely Depreciated Currencies," by R. A. Lehfeldt.

"Notes on Organisation for Promoting Industrial Efficiency," by S. de Smidt.

"Investigations on the Copper-Nickel Ores of the Rustenburg District," by J. A. Ortlepp.

September, 1922:—

"The Presidential Address," by G. A. Watermeyer.

"The Steel Age," by J. Henderson.

"Symposium: Miners' Phthisis," contributions by J. D. Marquard and R. A. H. Flugge-de Smidt.

October, 1922:—

"Symposium: Miners' Phthisis," contribution by C. J. Gray.

November, 1922:—

"Some Notes on the Evolution of the Konimeter," by R. A. H. Flugge-de Smit.

"Symposium: Miners' Phthisis," contribution by J. A. Woodburn.

"A New Method of Cleaning Vacuum Filter Leaves," by T. B. Stevens.

December, 1922:—

"The Value of Rand Gold Mining Investments," by W. A. Caldecott and G. J. V. Clarence.

"Alcohol Fuel Mixtures," by H. R. Adam.

January, 1923:—

"The Lowest Temperature yet Attained," by H. H. Paine.

"Return to Capital Invested in the Witwatersrand," by R. A. Lehfeldt.

"Symposium: Miners' Phthisis," contribution by J. Hayward Johnson.

February, 1923:—

"Recovery of Gold by Blanket Concentration in Substitution of Plate Amalgamation," by F. Wartenweiler.

"Notes on Cupellation Losses," by H. R. S. Wilkes.

"Note on the Contamination of Coke Samples when prepared in a Disc Grinder," by J. Henderson.

"Symposium: Miners' Phthisis," contribution by H. J. Ireland.

March, 1923:—

"The Solubility of Gold Amalgam in Cyanide Solution," by H. A. White.

"Observations on the Tin-Antimony-Lead Alloys," by R. W. Irwin.

April, 1923:—

"Further Note on Return to Capital Invested in the Witwatersrand," by R. A. Lehfeldt.

"Some Methods and Results in Dust-Sampling on the Witwatersrand," by A. Mavrogordato. (Abstract of Lecture.)

May, 1923:—

"Notes on Fan Ventilation on Deep Mines," by J. A. Woodburn.

"The Recovery of Gold Absorbed in Reduction Works," by A. King.

June, 1923:—

"The Use of Oxidisers in the Cyanide Treatment of Concentrates," by H. Rusden and J. Henderson.

"Symposium: Miners' Phthisis," Review of Symposium by A. J. Orenstein.

Technical Committees.—With the object of stimulating the interest of members in the work of the Society and securing an increased supply of original papers for reading, your Council has appointed Technical Committees to deal with the three branches of the Society's activities, viz.:—

Chemistry Committee, Mr. F. W. Watson, Chairman.

Metallurgical Committee, Mr. F. Wartenweiler, Chairman.

Mining Committee, Mr. C. J. Gray, Chairman.

These committees have busied themselves with getting into touch with possible authors, inviting suggestions regarding titles and subjects, and making arrangements generally with the view of obtaining papers. As a result of their efforts comprehensive lists of subjects have been drawn up, and already several promises to write papers in the near future have been received.

Research Endowment Fund.—The balance of this Fund is now £428 17s. 10d. Your Council has awarded Gold Medals to the following members for the best papers written during the year 1921-1922:—Messrs. R. A. Cooper, W. T. Heslop, E. P. Reim and F. W. Watson, and a Certificate of Merit to Mr. H. D. Bell. A full report of the presentation of medals and the certificate appeared in the October *Journal*.

Exhibition.—Your Council attempted to arrange for an Exhibition on a large scale to be organised in co-operation with the Johannesburg Chamber of Commerce and the Transvaal Manufacturers' Association. The project, however, did not commend itself to these institutions, and on this

account, together with the difficulty of obtaining a suitable building, it was decided not to hold an Exhibition this year.

Annual Dinner.—It has been decided to hold the Annual Dinner on the 21st July next, the reason for fixing the date so late being that it is desired to secure the presence of the Minister of Mines and Industries at the function. Mr. Malan has not been able to definitely promise to attend, but it is understood that he will do so if his Parliamentary and other duties permit.

Excursions.—By the courtesy of the Directors of the Companies concerned visits were made to the works of the Hume Pipe Company (South Africa), Ltd., on the 2nd September, 1922, and to the Vereeniging Brick and Tile Co., Ltd., and the Union Steel Corporation of South Africa, Ltd., on the 7th October, 1922. These visits proved highly interesting and instructive to those members who attended, though it is to be regretted that comparatively few took advantage of the opportunities offered. Through the kindness of the S.A. Institution of Engineers members were also privileged to take part in a visit to the works of the Rand Carbide, Ltd., the Shoes and Dies Branch of the Co-operative Smelting Works and the East Rand Engineering Co. at Germiston, on the 18th November last.

Arrangements have been made for a limited number of members to visit the Royal Mint and the works of the Cotton and Tobacco Exporting Co., Ltd., at Pretoria on the 31st July next.

Associated Scientific and Technical Societies of South Africa.—The Assessment levied by the Associated Societies for the year ending December, 1923, is the same as previously, that is, £1 1s. per head of members residing on the Witwatersrand and 10s. 6d. for those residing elsewhere in South Africa. As will be seen from the accounts the Society has had to sustain a loss of £33 13s. in respect of last year's assessment. It is hoped that a portion at least of this amount will yet be recovered. Your Council takes this opportunity of impressing upon members the necessity for the assessment being paid punctually, as the Society should not be called upon to meet deficits of this nature.

British Empire Exhibition, 1924.—The Society, as one of the Constituent Societies of the Associated Scientific and Technical Societies, has been included in a

general invitation to co-operate and assist in making up the South African exhibit. An announcement to this effect was made at the Ordinary General Meeting in March last.

War Memorial.—At the last Annual Meeting it was announced that arrangements were being made to place a memorial tablet in this hall in memory of those members and associates who lost their lives in the Great War. The Tablet has since been received and placed in position and was unveiled by Sir Lionel Phillips at a Special General Meeting held on the 10th February of this year. A full report of the ceremony was published in the February *Journal*.

Council.—Eleven meetings of the Council have been held during the year and several Committee Meetings. The attendance at Council Meetings averaged 12, the individual attendances being as follows:—

G. A. Watermeyer	10	A. King ¹	8
F. Wartenweiler		A. J. Orenstein ¹ ...	6
(Past President)	8	H. Pirow ⁶	6
H. R. Adam	10	J. E. Thomas ⁵ ...	1
C. J. Gray*	10	J. M. Thorburn ⁴ ...	6
F. W. Watson ²	8	J. Watson	8
J. R. Thurlow ²	9	A. Whitby	8
J. Chilton	4	H. A. White	5
R. A. Cooper ³	5	J. L. Willey ² ...	6
J. H. Johnson	5	J. A. Woodburn...	9

Leave of Absence granted for ¹ one meeting; ² two meetings; ³ four meetings; ⁴ elected July, 1922; ⁵ resigned October, 1922; ⁶ elected October, 1922.

The Council desires to place on record its appreciation of the services of the Hon. Auditors, Messrs. Alex. Aiken and Carter, and the Hon. Legal Advisers, Messrs. Van Hulsteyn, Feltham and Ford.

The President: In moving the adoption of the Council's Report and Accounts for the year it would not be out of place for me to make a few remarks on the progress of the Society during the year just ended. Following on my immediate predecessor Mr. Wartenweiler's suggestion with regard to technical committees furthering the interest in the meetings in catering for new original papers and stimulating interest in those presented by inviting discussion, we have been enabled to appreciate the wisdom and utility of the idea by the results of its working. We may view these results with a great deal of pride on account of the improvement in quality and number of the highly interesting papers with which we have been favoured. The adjudicators for

the medals will have a difficult task in deciding on the merits of the original papers read.

Our members who are not in the immediate vicinity of Johannesburg recognise that we appreciate their work and are heartened to further efforts by the knowledge that those at headquarters are eagerly on the look-out for their contributions. The advice of the President of the Institution of Mining and Metallurgy, London, Mr. R. Gilman Brown, in his recent presidential address to members on assumption of the chair should be taken to heart by members of all technical societies. He remarks that in order that a member may receive the full benefit and advantage from the privilege of membership he should give of his knowledge, no matter how small the contribution, as well as receive that of others. In this way only can the full advantage of membership be obtained. I would suggest, therefore, that any member having any knowledge to impart or who notices any error in contributions which he is in a position to discuss will come forward with his contribution to discussion and so provide for the dissemination of true knowledge. I should like to draw your attention to the fact that our Society is now in its thirtieth year. Its records show that on the 24th March, 1894, a meeting of metallurgists, chemists, and assayers was held at the North-Western Hotel with our old friend, Mr. A. F. Crosse in the chair, for the purpose of forming a society to be known as the Chemical and Metallurgical Society of South Africa. Its object was "for the purpose of bringing together members of the metallurgical and chemical profession and knowledge to discuss chemical questions for the general improvement of members of the knowledge of chemistry and metallurgy."

It will be generally admitted that the objects have been very successfully carried on by successive generations of members. It has been a power for good not only in South Africa but throughout the technical world. We have traditions to look back upon which should stimulate us to further efforts. These traditions must be maintained. It is the duty of the present members to assume the mantle of their predecessors and prevent its dragging in the dust. You will notice from the report that the membership has apparently fallen from 570 in 1922 to 529 in 1923. Since we have joined the Associated Scientific and Technical Societies we have assumed responsibility of an assessment of £1 ls. per

head in respect of Witwatersrand members. It will therefore be seen that it becomes impossible for the Society to carry names on its roll if the subscriptions are not forthcoming, since each defaulting member becomes a pecuniary loss to the Society not merely by his default of his subscription but by the rendering of the Society liable for his portion of the assessment. It may be mentioned that a very careful scrutiny of the roll is necessary to eliminate direct loss. The percentage of subscription payers has been increased to 84 per cent., indicating a very healthy state of the finances. In taking leave of the chair, I should like to tender my congratulations to the Society on the choice of my successor. Mr. Fred Watson has for many years been a tried and valued member of the Council, and will undoubtedly be an ornament to the chair. He has always placed the interests of the Society before his own convenience. He has a very high reputation in his profession and the conduct of the representation and business of the Society may be safely left in his hands.

To those associated with me during the past year as fellow members of Council I beg to tender my thanks for the courtesy shown me, and the support I have received from them during my term of office. To Mr. Thurlow I am sure you will allow me to tender the thanks of the Society in general, and myself in particular, for the sterling qualities he has displayed in guiding the financial side of our business.

I should also like to place on record my admiration of the very able manner in which our Secretary, Capt. Jeffreys, and his staff have carried out their duties during my term of office.

With these few remarks I beg to move the adoption of the report of Council, which is now before you.

Mr. F. W. Watson had very much pleasure in seconding the adoption of the Annual Report. He had nothing further to add at the moment as he would have some remarks to make under another matter later on.

Mr. John Watson said he would like, as an ordinary Member of the Council and a member of the Finance Committee, to support the adoption of the Annual Report. He would like also to endorse very strongly the remarks made with reference to their Hon. Treasurer and to Capt. Jeffreys.

THE CHEMICAL, METALLURGICAL AND MINING SOCIETY OF SOUTH AFRICA.

Dr.	REVENUE AND EXPENDITURE ACCOUNT for Twelve Months ended 31st May, 1923.				Cr.	
	1921-22. £ s. d.	1922-23. £ s. d.	1921-22. £ s. d.	1922-23. £ s. d.	1921-22. £ s. d.	1922-23. £ s. d.
To CHARGES (including Post-ages, Bank Charges, Insurance, &c.)	81 16 10	115 3 3	6 30 11 0	616 0 0		
PRINTING AND STATIONERY	120 2 2	81 12 7	46 4 0	73 10 0		
RENT, LIGHTING AND TELEPHONE	128 18 2	132 3 4	201 17 3	179 11 0		
SALARIES	680 7 5	600 0 0	17 6 6	32 10 6		
			0 10 6	0 10 6		
GRANTS AND PRIZES—		988 19 2	886 9 3	902 2 0		
Vice Chancellor's Prize, Witwatersrand University			36 15 0			
Grant to Seymour Memorial Library	25 0 0	25 0 0				
Prizes—Witwatersrand University	3 3 0					
ASSOCIATED SCIENTIFIC AND TECHNICAL SOCIETIES OF SOUTH AFRICA—						
Assessment for 1922		271 8 6	(544 5 9)	603 16 4		
Less Collected from Members		237 15 6	383 5 0	387 12 1		
			161 0 9	216 4 3		
COST OF WAR MEMORIAL TABLET		33 13 0				
TRANSACTIONS—						
Sales		34 11 5	24 5 3	41 10 3		
Cost of Binding			11 0 0	27 1 0		
Less Sales			6 0 0			
BALANCE, being excess of Revenue over Expenditure	1 9 10	108 7 11				
	84 12 10					
	£1,125 10 3	£1,195 11 6	£1,125 10 3	£1,195 11 6		

BALANCE SHEET as at 31st May, 1923.

Dr.

1921-22.	1922-23.	1921-22.	1922-23.	Gr.
£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d. £ s. d.

LIABILITIES.

162 16 3	To Sundry Creditors	38 11 5		
25 0 0	" SEYMOUR MEMORIAL LIBRARY ...	25 0 0		
2 2 0	" SUBSCRIPTIONS PAID IN ADVANCE—			
3 13 6	Members... ..	24 13 6		
	Associates	3 13 6		
412 15 10	" RESEARCH ENDOWMENT FUND ...	28 7 0		
(350 13 1	" REVENUE AND EXPENDITURE A/c—	128 17 10		
84 12 10)	Balance at 31st May, 1922 ...	435 5 11		
	Add Excess of Revenue over			
	Expenditure for year ended			
435 5 11	31st May, 1923	108 7 11		
£1,041 13 6		543 13 10		
		£1,064 10 1		

We have examined the above Balance Sheet and accompanying Revenue and Expenditure Account together with the Books and Vouchers of the Society for the year ended 31st May, 1923, and have obtained all the information and explanations we have required. In our opinion such Balance Sheet is properly drawn up so as to exhibit a true and correct view of the state of the affairs of the Society at the 31st May, 1923, according to the best of our information and the explanations given to us, and as shown by the Books of the Society.

JOHANNESBURG, 14th June, 1923. (Sd.) ALEX. AIKEN & CARTER, Auditors.

RESEARCH ENDOWMENT FUND. REVENUE AND EXPENDITURE ACCOUNT for the Year ended 21st May, 1923.

Dr.	1921-22.	1922-23.	1921-22.	1922-23.	Gr.
£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.
7 5 0	To EXPENSES—				
	Collins Bros., for Four Gold				
	Medals for year 1922	7 5 0			
	Horters, Ltd., Cost of Embos-				
	sing 50 Certificates	1 12 6			
	Bank Charges... ..	0 0 6			
	BALANCE, being Excess of	8 18 0			
	Revenue over Expenditure for				
17 15 0	year ended 31st May, 1923...	16 2 0			
£25 0 0		£25 0 0			
		£25 0 0			

BALANCE SHEET AS AT 31st May, 1923.

1921-22.	1922-23.	1921-22.	1922-23.	Gr.
£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.
395 0 10	To ACCUMULATED FUNDS—			
	Balance at 31st May, 1922 ...	412 15 10		
	Add Excess of Revenue over			
	Expenditure for the year			
17 15 0	ended 31st May, 1923	16 2 0		
492 15 10		492 15 10		

ASSETS.

26 6 1	By CASH—			
	Standard Bank of South			
9 15 3	Africa, Ltd., Johannesburg	21 4 5		
2 1 5	Standard Bank of South	68 16 10		
	Africa, Ltd., London... ..	2 0 10		
405 10 10	On hand			
	Standard Bank of South			
	Africa, Ltd., Johannesburg			
	On Fixed Deposit with	405 7 10		
	Accrued Interest	497 9 11		
	SUNDRY DEBTORS—			
100 0 0	Advertisers	91 12 4		
40 4 1	Stundries	1 10 0		
45 0 0	OFFICE FURNITURE	93 2 4		
412 15 10	TRUSTEES "RESEARCH ENDOW-	45 0 0		
	MENT FUND"	428 17 10		
£1,041 13 6		£1,064 10 1		

(Sd.) J. R. THURLLOW.

Hon. Treasurer.

(Sd.) H. A. G. JEFFREYS.

Secretary.

their Secretary. Their report, from the financial point of view, was the best they had had for some years. Very great credit was due to their Hon. Treasurer and to their Secretary for the very able way in which they had looked after the economic side of the Society's affairs.

The adoption of the Annual Report and Statement of Accounts was passed unanimously.

The Secretary read the report of the Scrutineers on the declaration of the result of the ballot for Officers and Council for the ensuing year, as follows:—

Johannesburg,
23rd June, 1923.

The Chairman,
Annual General Meeting,
Chemical, Metallurgical and Mining
Society of S.A., Johannesburg.

Dear Sir,—

*Election of Officers and Council for the Year
Ending June, 1924.*

We have to report that we have inspected the Nomination Papers and received ninety-three envelopes containing Ballot Papers, and as the result of our scrutiny we find that the following gentlemen have been elected:—

PRESIDENT: Mr. F. W. Watson (unopposed).
VICE-PRESIDENTS: Messrs. F. Wartenweiler,
H. A. White and C. J. Gray.

HONORARY TREASURER: Mr. J. R. Thurlow (unopposed).

MEMBERS OF COUNCIL: Dr. A. J. Orenstein, Mr. H. R. Adam, Mr. J. A. Woodburn, Mr. J. Chilton, Mr. R. A. Cooper, Mr. M. O. Tillard, Mr. J. Watson, Mr. J. M. Thorburn, Mr. A. King, Mr. J. H. Johnson, Mr. J. L. Willey, Mr. A. Whitby.

Yours faithfully,
P. T. B. MORRISBY,
J. M. DIXON,
G. MELVILL,
Scrutineers.

Mr. F. W. Watson said he had to thank the members of the Society for the great honour they had done him in electing him as President for the ensuing year. The retiring President, in his valedictory address, had had a lot of nice things to say about his (the speaker's) services to the Society, which, after all, had been very small. He was pleased to think the Society had had such a successful year, and would like to endorse the remarks of Mr. John Watson with reference to the work of the Finance Committee. He thought it was a most satisfactory state of affairs, after the many lean years the Society had experienced, to think they could now be considered on a pretty good financial basis. He thought also the

Society was greatly improved in that it was not so difficult to get papers as it was a few years ago, when they used to have to scramble round for them.

He wanted to draw their attention to one or two things. The first was they were having an Annual Dinner next month, and he hoped members present at, as well as those who were absent from the meeting—who would no doubt hear about it through circulars to be issued—would support the function which had been allowed to lapse now for some three or more years. Before the war it used to be an annual event, but since then it had been allowed to lapse to a large extent. They were now, however, in the happy position that they had not to go into the high-ways and by-ways to look for a hotel; they had their own club house. He thought members of the Society ought to support the function, so that Members of Council could go forward in the future and have it every year instead of letting it lapse on account of loss.

He hoped they would maintain the position they had now reached in the way of papers. He felt every confidence, because those Technical Committees, on the proposition of Mr. Wartenweiler, had all met, and there had been a great many suggestions put forward. If only a percentage of these came to fruition they would do very well as regards papers.

He also wished to thank members for the very excellent Office Bearers and Council they had elected for the ensuing year. He knew he would have their most hearty support. He also would like to express his appreciation of their Secretary, Mr. Jeffreys, and also the valuable work Mr. Thurlow had done as Hon. Treasurer of the Society. In conclusion, he would again thank members for the honour they had done him in electing him President.

Mr. C. J. Gray, on behalf of the Vice-Presidents, thanked members for electing them. He could safely promise that they would all do their best to support Mr. Watson in the Presidential chair. It gave one pleasure to be a Vice-President of a Society which, at the present time, at least, was a very live society. There was no difficulty, as Mr. Watson had indicated, with regard to the supply of papers, and that was satisfactory, for the whole object of the Society was to spread and exchange knowledge. Those papers which they were so fortunate as to obtain not only gave information in

themselves but they led to discussion in which members might give information in exchange, and in which, as Prof. Watermeyer had said, errors could be drawn attention to and the truth made plain.

Dr. Orenstein, on behalf of the Council, thanked members for their election. For himself, he wanted to thank them for having elected him for the fourth time to the Council. It was always a great pleasure to him to serve on the Council and to assist in forwarding the interests of the Society. Perhaps he was not in a position personally to do as much for the bulk of the members of the Society in a scientific way as some of those who were working in the technological fields with which the Society dealt; but it had been a great pleasure to him, and he considered it a great honour to be on that Council. He thanked them all for the honour they had done them in electing them. Before he sat down, he would ask members present to pass a hearty vote of thanks to the gentlemen who so kindly went through the ballot papers.

Mr. J. R. Thurlow seconded the vote of thanks to the Scrutineers, and took the opportunity of thanking members for the honour they had done him in re-electing him as Honorary Treasurer. The President and other members had given him a lot of praise in connection with his handling of the finances. While dealing with the matter of finance, the accounts section of the Annual Report, which had been read, explained the position very fully, and there was not very much he could add to it. They would have noted that the General Expenses were slightly higher, in spite of their best efforts to keep them down; but they would doubtless recall that at the last Annual General Meeting a suggestion was put forward that they should issue a list of members. Well, that was done during the year, at a cost of £21. He thought they might call that a non-recurrent charge. There was also the War Memorial, the landed cost of which was £34 odd. Additional expenses were incurred in that connection; for instance, in having the tablet placed in position, printing and issuing cards of invitation, the Special General Meeting for the unveiling ceremony, and so on. He thought they might be quite satisfied that no undue extravagance had been incurred. The revenue from the *Journal*, which was one of their principal sources of revenue, was appreciably better than for the previous year. They made a profit of about £161 last year;

this year they had made £216. He would like to mention that, as the accounts showed, they had lost some £33 due to certain members not having paid up their assessment fees due to the Associated Scientific and Technical Societies. While the Society had been able to meet the assessment in full this year, he thought it was very wrong in principle that the Society should have to draw on its funds for that sort of thing, and he could only trust that it would not happen again and that everyone would meet them in the matter.

On behalf of the Finance Committee he could only emphasise what their President had said about the good work of their Secretary. Those of them who had worked very closely in touch with him knew perfectly well that it was his close attention to detail and marked financial ability which was the reason that they had done so well in the past year. So far as he, personally, was concerned, as head of the finances, he had had a very easy job; he had simply been on hand to sign things, and that sort of thing. It was really due to their Secretary.

Mr. P. T. Morrisby, on behalf of the Scrutineers, had much pleasure in thanking members for the very hearty vote of thanks they had accorded them that evening. He could assure them that the Scrutineers' task was a great pleasure. Their only regret was that the number of voting papers was not more, and they hoped that, in the coming year, with the enrolment of new members, it would be a more arduous duty. He regretted very much that one of the Scrutineers was absent that evening, namely, Mr. Newton, and they would be sorry to hear that he was on the sick list.

He looked upon the meeting of Scrutineers as the birthday anniversary of the Society. As they sat round the Scrutineers' table he could not but help remarking to the Secretary and the Scrutineers: "I wish you many happy returns of the day." He wished the Society many happy returns of the day. He trusted that the new President would be well supported, that they would have a large enrolment of members, and that the work of the Society would be carried along to a still greater degree of success.

GENERAL BUSINESS.

ANNUAL DINNER.

The President directed the attention of members to the Annual Dinner of the Society on the 21st July next, whereat he hoped as

many members as possible would be present. He would also like members to notify the Secretary as early as possible as to whether members would bring their ladies.

VISIT TO ROYAL MINT, PRETORIA.

The President also drew attention to the proposed visit to the Royal Mint at Pretoria, on the 31st July, and to the Works of the Cotton and Tobacco Exporting Co., Ltd., Pretoria, on the same day.

SOLUBLE PARTICLES IN KONIMETER SPOTS.

Mr. R. A. H. Flugge-de Smidt read the following note:—By continuing the process, referred to last month, of placing a drop of some solvent directly on a konimeter spot and using a radiator as a source of heat as well as light, one can not only observe the edge of the contracting film pass over the spot, but also watch the liquid dry up completely.

Several spots taken in a fog, caused by a machine while drilling, were treated in this manner, and before the liquid dried up crystals were seen to form. By slowing down the final process through cooling, the crystals can be grown to a fair size, and they sometimes assume a recognisable shape. Some observed by the writer attained a length of 200 microns and a breadth of 20 microns, with re-entrant angles at each end.

It is easy to test the solvent, whether distilled water or dilute HCl for impurities, by drying up a spot of the same beforehand.

To emphasize the results, sometimes 100 c.c. of air was sampled, instead of the usual

5 or 10 c.c. All spots were taken on a plain glass slide, and often needle-shaped crystals up to 50 microns in length were observed on such spots before treatment.

On one occasion the original spot showed a number of rod-like crystals 20 microns in length. HCl vapour had no effect on these even when the cover-glass was left in position for twenty-four hours. A drop of distilled water dissolved them, and they only re-crystallised when the merest trace of water was still left. Presumably the salt was very soluble in water.

Care must be taken, of course, that the drop of liquid used must be so small that, when flattened out under the fragment of cover-glass, no portion of the liquid comes in contact with a contaminated portion of the slide.

Contamination of Slides.—Mention was made last month of contamination of slides noticed during the recent test carried out by the Chamber of Mines. The writer offers a possible explanation.

Slides prepared with glycerine jelly would in a week's time become unfit for use—the jelly would dry up and crack, and there would be innumerable points to reflect light when observed with a dark background. The thinner the film the quicker the process.

To test the idea a film that had been prepared and had not left the laboratory was observed two days later, and showed a highly contaminated field. Some mines in the Far East Rand could only send in their slides for examination on the second day.

THE USE OF OXIDISERS IN THE CYANIDE TREATMENT OF CONCENTRATES.

By H. RUSDEN and J. HENDERSON.

Black sand and miscellaneous concentrates (resulting mostly from the clean-up of old Reduction Works sites) are amongst the by-products received at the Witwatersrand Co-operative Smelting Works.

Clean current black sand usually yields a reasonably satisfactory extraction, but frequently fouls the cyanide solutions, charging them heavily with reducing matter.

Miscellaneous concentrates are mostly of a much more refractory character, and con-

signments received from the same source vary considerably from the gold extraction standpoint in the cyanide plant.

This irregularity makes it necessary to conduct a preliminary experimental extraction test on all lots of material that may be suitable for cyanide treatment.

The Supervisors of the Smelting Works suggested the use of oxidisers might be beneficial and that a trial be made first on an experimental basis.

The usual works' experimental extraction test was used in connection with the various oxidisers, and is conducted as follows:—

The sample is dry crushed to —200 mesh (aperture 0.003in.). One hundred grams are weighed off, together with 2 grams of powdered lime and the mixture run into a clean screw-stoppered bottle (not less than $2\frac{1}{2}$ litre capacity), into which has been previously measured 100 c.c. water (Rand Water Board), making the ratio 1 to 1. The bottle is given a rapid shake round to prevent the mixture forming clots. Then about 130 grams of mercury are added, and the bottle is agitated for one hour. For agitating, we use an ordinary belt pulley, 32in. diameter, fitted with clamps. The bottles are securely clamped to the spokes, and the pulley will take eight bottles, four on each side. The revolutions of the pulley are 18 per minute. After one hour's agitation the pulp is poured into a 9in. diameter porcelain dish and the bottle thoroughly rinsed into the dish. The pulp is then washed back into the bottle with a stream of water, taking care no mercury goes over the edge of the dish. As soon as the mercury is visible it is carefully examined to see if any flouring has taken place, and, if so, the fact is noted in the experimental extraction book. Any floured mercury, as far as possible, is gathered into the main body by rubbing round with a pestle. The mercury is then completely separated from the remaining slime by pouring from one dish to another. The bottle stands till the slime has settled then the clear liquid is syphoned off down to a mark on the bottle which indicates that there are 250 c.c. of liquid remaining, making the ratio $2\frac{1}{2}$ to 1. A further half gram of lime is added, and then 10 c.c. of strong cyanide solution (about 40 grams sodium cyanide per litre). This makes the solution in the bottle equal to about 0.15 per cent. to 0.20 per cent. K Cy. The bottle is agitated again for twelve hours. A small quantity of the pulp is poured into a porcelain dish and notes made as to the presence of froth or grease. Ten c.c. are filtered and tested for cyanide and alkalinity. If the cyanide is reduced to under 0.04 per cent. a further 10 c.c. of strong cyanide is added and the bottle again agitated for another twelve hours. If we suspect heavy cyanide consumption from previous experience of material from the same source, 20 c.c. of strong cyanide are added, making the solution in the bottle equal to 0.30 per cent. to 0.40 per cent. K Cy. In using 2 grams

of lime at the commencement of the test, and an additional half gram just before adding the cyanide, we find the alkalinity never excessive and never insufficient. We also test for reducing matter present in the solution by acidifying 10 c.c. with dilute sulphuric acid and titrating with decinormal permanganate of potash. The results are expressed in terms of milligrams of oxygen consumed per litre. Remarks on this test will be made further on in the paper. The bottle is allowed to stand for three days with the stopper out, and given a shake each day. It has been found that precipitation of dissolved gold sometimes takes place when the solution is left in contact with the slime, hence the three days' contact to imitate the working scale practice as near as possible.

At the end of three days the pulp is poured into an amalgamated copper prospecting pan and the bottle completely washed out into the pan. Then the pulp is washed out of the copper pan into an ordinary 18in. prospecting pan and the latter filled with water to wash the residue. The object of the copper pan is to retain any small globules of mercury or amalgam that might be in the pulp. As soon as the slime has settled the clear liquid is poured off, and the residue drained as much as possible. It is then dried and sent to the assayer. From the gold in the residue the percentage of extraction is calculated.

This experimental extraction test, besides furnishing information as to the gold extraction, supplies a guide as to the probable cyanide and lime consumption, also when excessive frothing may be expected in practical treatment. The results in the Works correspond very closely with the experimental tests.

In the experimental trials with various oxidisers, none of the details of the test just described were altered, the oxidisers being additions to it, and were added after amalgamation just before the addition of the cyanide.

Permanganate of potash was first tried. A black sand which had shown, experimentally, a very poor yield, giving a residue containing 33.0 dwts. gold and an extraction of 89 per cent. was chosen. Permanganate, nearly equal to the amount required to oxidise the reducing matter (equivalent to 4.4lbs. per ton of dry material) was added after amalgamation, but before the addition of cyanide. The bottle was well shaken and then the

TABLE "A."

Reference No.	Material.	Original Gold Assay. Ozs.	Strength Cyanide Solution		Alkalinity.	Pernanganate test expressed in M'grams Oxygen consumed per litre.	Oxidiser added equal to lbs. per ton of material.	Washed residue Gold dwts.	Extraction %.	REMARKS.
			Before Agitation.	After Agitation.						
1.	Black Sand	15.4	0.36%	0.075%	0.022%	152	NIL	33.0	89.0	
	"	"	0.36	0.085	0.015	112	4.4 Permanganate	3.0	99.0	
	"	"	0.36	0.175	0.032	144	20.0 BaO ₂	6.0	98.0	
	"	"	0.19	0.040	0.028	160	20.0 PbO ₂	9.0	96.3	
2.	Black Sand	8.6	0.19	0.060	0.012	144	NIL	11.0	93.6	
	"	"	0.19	0.090	0.010	136	3.4 Permanganate	6.5	96.2	
	"	"	0.19	0.075	0.014	120	10.0 BaO ₂	5.0	97.1	
	"	"	0.19	0.050	0.019	128	10.0 PbO ₂	5.0	97.1	
3.	Black Sand	5.5	0.18	0.055	0.015	88	NIL	12.0	89.1	An example of where Oxidisers had practically no effect.
	"	"	0.18	0.050	0.015	56	1.58 Permanganate	10.0	90.9	
	"	"	0.19	0.070	0.019	112	20.0 BaO ₂	13.0	88.1	
4.	Misc. Conc.	10.1	0.19	0.122	0.074	56	NIL	19.5	90.3	The results with BaO ₂ were so different to the other Oxidisers, it was repeated with confirmatory results.
	"	"	0.19	0.120	0.071	64	8.2 Permanganate	20.0	90.1	
	"	"	0.19	0.120	0.077	52	10.0 BaO ₂	5.5	97.2	
	"	"	0.19	0.100	0.082	48	10.0 PbO ₂	28.0	86.1	
5.	Black Sand	13.9	0.38	0.095	0.036	104	NIL	9.0	96.7	In this experiment mixtures of the Oxidisers were tried with no better results. After this experiment Permanganate and only PbO ₂ were dropped and only BaO ₂ used in further trials.
	"	"	0.38	0.100	0.037	128	4.4 Permanganate	9.0	96.7	
	"	"	0.38	0.115	0.036	112	10.0 BaO ₂	7.0	97.4	
	"	"	0.38	0.075	0.047	104	10.0 PbO ₂	8.0	97.1	
	"	"	0.38	0.105	0.042	112	(1.4 Permanganate) (3.4 BaO ₂) (3.4 PbO ₂)	10.0	96.3	
	"	"	0.38	0.110	0.042	116	(2.2 Permanganate) (5.0 BaO ₂)	7.0	97.4	
6.	Misc. Conc.	9.8	0.38	0.165	0.084	52	NIL	26.0	A 86.7	A and B were tried first then C and D with increased amounts of BaO ₂ . This showed that 10.0 lbs. or .5% was not enough. From this experiment 15 lbs. or .75% was used in further trials.
	"	"	0.38	0.180	0.098	52	10.0 BaO ₂	24.0	B 87.7	
	"	"	0.38	0.170	0.108	48	20.0 BaO ₂	6.5	C 96.6	
	"	"	0.38	0.170	0.104	48	15.0 BaO ₂	7.0	D 96.4	
7.	Misc. Conc.	8.2	0.19	0.115	0.074	29	NIL	48.0	70.7	An example of a big increase in the extraction with BaO ₂ .
	"	"	0.19	0.135	0.106	38	15.0 BaO ₂	8.5	94.8	
8.	Misc. Conc.	15.3	0.175	0.055	0.031	43	NIL	16.0	94.7	Examples of satisfactory improvement with BaO ₂ .
	"	"	0.175	0.065	0.060	43	15.0 BaO ₂	5.0	98.3	
9.	Misc. Conc.	17.3	0.175	0.100	0.016	33	NIL	24.0	93.0	An example of the Reverse effect with BaO ₂ .
	"	"	0.175	0.105	0.045	33	15.0 BaO ₂	6.0	98.2	
10.	Misc. Conc.	26.2	0.380	0.170	0.047	33	NIL	25.0	95.2	
	"	"	0.380	0.115	0.060	33	15.0 BaO ₂	58.0	88.5	

TABLE "B."

Charge No.	Average charge Assay, Gold, Ozs.	EXPERIMENTAL EXTRACTION TESTS.				WORKS RESULTS.		
		Without Oxidiser.		With BaO ₂ .		BaO ₂ used, lbs. per ton.	Residue.	
		Residue Dwts.	Extraction %.	Residue Dwts.	Extraction %.		Gold Dwts.	Extraction %.
303	9.9	23.7	88.0	6.0	96.9	20.0	8.0	96.0
306	10.0	30.0	85.0	11.4	94.3	18.5	12.0	94.0
311	14.3	32.6	88.6	8.6	97.0	20.0	18.0	93.7
313	15.4	18.2	94.1	11.7	96.2	15.0	10.0	96.7
316	13.1	39.6	84.9	5.5	97.9	15.0	8.5	96.7
352	23.5	28.0	94.0	14.0	97.0	15.0	13.0	97.7
355	12.3	15.0	94.0	7.1	97.1	15.0	6.0	97.5
363	13.4	18.0	93.2	8.0	97.0	15.0	7.0	97.4

cyanide added. The test was conducted in the usual way and the residue assayed only 3.0 dwts., giving an extraction of 99 per cent. This was quite a startling result, and on being brought to the notice of the supervisors, instructions were given to make further experiments with permanganate of potash, lead peroxide and barium peroxide. A selection of the results are given in Table A, Nos. 1 to 5, together with other results using barium peroxide only, and may be summarised as follows:—

All three gave generally improved extraction; permanganate sometimes slightly increased cyanide consumption, lead peroxide invariably increased cyanide consumption, but barium peroxide usually decreased cyanide consumption. All three at times gave disappointing results. It is to be hoped that discussion may throw some light on the possible reason why the results obtained when using oxidisers at times yield a worse extraction than when no oxidiser is added.

Barium peroxide on the whole gave more consistent results than permanganate or lead peroxide, and instructions were given to make all further experiments using barium peroxide only.

Experiments with varying proportions of barium peroxide were tried, from 1 per cent. of dry weight of concentrate to 0.5 per cent. These experiments showed that, on the whole, 0.75 per cent. gave satisfactory results and was therefore adopted for all further experiments (see Table A, No. 6).

The next step was a practical trial. A charge of 10 tons was made up of various lots which had experimentally showed a good extraction with the aid of barium peroxide. This charge was milled and two hundred pounds of barium peroxide were spread over the settled charge in the collector, just before being transferred to the "Brown" tank for agitation. Cyanide was then added and agitation carried on in the usual way. The results are shown in Table B (Charge No. 303), together with several other barium peroxide charges. Generally speaking, the practical trials agree very closely with the experimental, and must be considered satisfactory when it is noted that in the experimental trials a clean cyanide solution is used, but in the practical, a Works solution that contains a considerable quantity of reducing matter. We cannot say that barium peroxide can be used always with success with any concentrate. From November, 1922, to April, 1923, we have tried experimentally 68 consignments of concentrates with barium peroxide, with the following results:—

Samples giving a satisfactorily increased extraction, 33, or 48.5 per cent.

Samples giving increased extraction, but not sufficient to warrant practical trial, 20, or 29.4 per cent.

Samples giving a worse result than without the oxidiser, 15, or 22.1 per cent.

Those giving satisfactorily increased extraction were made up into charges and treated

in the cyanide plant with barium peroxide, with good results.

The others were treated by furnace methods (pan furnace or sinter pots, with subsequent treatment in the blast furnace). We have adopted the following method of procedure in dealing with material that may be treated in the cyanide plant.

Any sample, which on the experimental test gives a poor extraction without barium peroxide, and which in other respects is a desirable material for the cyanide plant, is put through a second experimental trial using 0.75 per cent. barium peroxide. If the result is satisfactory that particular consignment is reserved until sufficient material which has been proved to be satisfactory with the oxidiser has accumulated to make up a full charge of 10 tons.

Formerly, roasting concentrates was resorted to, when experiment showed that improved extraction was obtained thereby, but it is much easier and less expensive here to use barium peroxide; in fact, our roasting furnace has been dismantled. One undoubted advantage of barium peroxide is that any which enters the solution, precipitates soluble sulphates without adding to the solution any further soluble salt (see *Rand Metallurgical Practice*, Vol. 1, page 384, note by Dr. Caldecott).

With regard to the permanganate test for reducing matter; we find it a useful guide in the cyanide works as to the degree of "fouling" of the solutions which we may expect, and bleaching powder is added accordingly except when barium peroxide is used. Another interesting point is that the addition up to as much as 1 per cent. oxidiser to most of the samples experimented on did not materially affect the reducing matter present as determined by the permanganate test.

One particular black sand always gives us in the experimental extraction tests a reducing matter figure three times the average of the other black sands. It also consumes more cyanide and lime than the average. But the extraction is generally satisfactory. A charge of this material in the cyanide plant gave the following results in oxygen present in the solution after agitation (we sample our "Brown" tanks just at the end of agitation):—

After first 12 hours' agitation, dissolved oxygen $\frac{1}{2}$ milligram per litre.

After second 12 hours' agitation, dissolved oxygen 1 milligram per litre.

After third 12 hours' agitation, dissolved oxygen $\frac{3}{4}$ milligram per litre.

Our solutions were at this time very foul. The figure given by the permanganate test for reducing matter was equal to a consumption of 912 (nine hundred and twelve) milligrams per litre. Yet the extraction was good. The original assay was 12.7 ozs. No barium peroxide was used. The final residue was 8.0 dwts=96.8 per cent. extraction. The above is typical of our experience with this particular black sand.

In view of Elsner's equation, which includes oxygen as an essential for the solution of gold in cyanide solutions, the above figures are interesting.

These experiments were made under the direction of Dr. Caldecott and Messrs. Maxwell and Pearce.

The assays were made by Mr. A. L. Wright, Smelting Works' assayer.

Our thanks are due to the Witwatersrand Co-operative Smelting Works, Ltd., for permission to publish this paper.

Mr. H. A. White had very much pleasure in expressing the thanks of members to the authors for the most illuminating and interesting paper on oxidisers that they had laid before the Society for very many years. Generally speaking, amongst metallurgists the use of oxidisers in cyanide works was looked upon as the exploitation of an exploded theory; they felt sure from the results of experiments that they had made in the past, and from experiences that they had had, that no good could come out of that particular Nazareth. But circumstances altered cases, and where they had charges carrying ounces of gold per ton in place of the pulp in the ordinary cyanide works carrying pennyweights per ton, oxidisers had perhaps a profitable field where it would be utterly impossible to make them successful in current practice.

Some of the experimental results were distinctly baffling, and no doubt the authors had found that to be the case: take for instance barium peroxide; in one case they got an increased extraction of 5 or 10 per cent., and in another case there was a reduced extraction; and they wanted to know why. He did not propose to solve the problem, but he had no doubt the direction in which the explanation could be sought was in this way. The oxidiser, of course, not only functioned to provide the necessary oxygen as required by Elsner's Equation, but it also had a further powerful effect upon

certain sulphides that might be present. Some years ago he made a series of experiments on the sulphides found commonly on the Witwatersrand. He found that pyrite was very, very difficult to oxidise in ordinary treatment, that marcasite was very much more easily oxidised, and what was more, it gave very much more deleterious products. Marcasite (FeS_2) in itself, did not do very much harm, but when it was oxidised to form FeSO_4 , the first effect of adding an oxidiser was to make things a little bit worse than before. There was another pyritic ore present, that was pyrrhotite, especially on the East Rand pyrrhotite was very frequently found. In the presence of oxidisers pyrrhotite was attacked rapidly with formation of thiosulphates, even in the presence of cyanide thiosulphate was formed first, and afterwards slowly converted to thiocyanates. In that direction might be found possibly the explanation of some of the contradictions revealed in the table before them.

There was one other remark he would like to make, that was in reference to the use of the term "foul solution." It was a use he had deprecated before in that Society; he did not like the words "foul solution." What was meant, he supposed, was an oxygen consuming solution containing oxygen absorbers in a particularly active form. Principally, he supposed, the consumer of permanganate in a test of their solutions would be sulpho-cyanides, therefore a large discount must be made from any figures got by taking note of the sulpho-cyanides present, which should be independently determined. Further, in all those solutions, especially where they dealt with pyritic material, they had thiosulphates present; and, in considerable quantity, no doubt they might be deleterious.

He was rather sorry they did not have a detailed analysis of some of those solutions, especially in one case where barium peroxide was of advantage, and in another case where it was of disadvantage.

There was one other point he would like to make. He was glad to notice that when adding permanganate they had apparently done it before adding the cyanide, because in many cases tests had been made of permanganate when it had been added simultaneously with the cyanide. In that case, of course, the destruction of cyanide was very nearly as rapid as of organic reducers present. No permanganate remained, and the fact was they were really experimenting with colloidal manganese dioxide. Probably

members of the Society would remember that a few years ago they had a paper by Mr. Adair, in which he advocated strongly the use of umber, and he recorded in his paper some very remarkable results, results even better than those displayed on the board that night. He would not detain members any further, except to move a hearty vote of thanks.

Mr. A. King, in seconding the vote of thanks, mentioned that this paper, following that of Mr. Wartenweiler's on Corduroy Concentration, came none too soon, since with the concentration methods now so largely in use considerably greater quantities of "black sand" were being produced, which in some cases at least required separate treatment. Mr. White's recent paper on "The Solubility of Gold Amalgam in Cyanide Solutions" also had an indirect bearing on this subject, because it had been his experience that where steps were taken to eliminate all amalgam from black sand before cyanide treatment more satisfactory results were obtained in a shorter time.

It would be of interest to many members to get the details of the preliminary tests given to each consignment of concentrate before the method of treatment was determined, and the value of these investigations would be realised.

On the necessity for the use of oxidisers opinions must differ, and it is only by thorough investigations such as those carried out by the authors that a decision can be arrived at. An article recently appeared in the technical press* which showed as a result of a series of parallel tests, the efficacy of sodium peroxide in the cyanide treatment of a telluride ore, and the author of that article thought it probable that similar improvements might be effected by using sodium peroxide in the cyanide treatment of concentrates. Messrs. Rusden and Henderson have shown that such is often the case.

Regarding the consumption of oxygen during the air-agitation of concentrate, this may be partly due to the presence of alkaline sulphides, but may also be caused by fine iron, and it would be interesting to know whether the authors have experienced any difficulty from this source, and whether the excessive consumption of oxygen in the case cited in the paper might not have been caused by fine iron.

* "The Use of Sodium Peroxide in Cyanidation at the Teck-Hughes Gold Mine."—*Eng. & Min. Journal Press*, March 10, 1923.

SUMMARY OF CONTRIBUTIONS TO MINERS' PHTHISIS SYMPOSIUM.

By A. J. ORENSTEIN.

The Symposium on Miners' Phthisis was opened at the 19th March, 1921, meeting, and the Society is to be congratulated on the quality as well as the quantity of the contributions made. The total number of contributions aggregate twenty-seven. The final one was made in April of the present year.

The majority of the contributors dealt with the prevention of miners' phthisis from the standpoint of measures to be taken underground. It is noteworthy that the importance of adequate ventilation was stressed by the majority of contributors.

Ventilation.—Of the contributions dealing with the importance of ventilation the report of the Joint Committee of the Government Mining Engineer's Department and the Chamber of Mines, read by Mr. C. J. Gray at the October, 1922, meeting, is particularly interesting. In this report it is pointed out that water blasts do not wash out fine dust, that this may remain suspended in the air for many hours in spite of the water blast; and a special point is made of the fact that the dangerous dust can only—and must be—removed by ventilation.

Mr. J. A. Woodburn at the November 22nd, 1922, meeting supplemented this dictum by reading extracts from a paper read before the Australasian Institute of Mining and Metallurgy, in which it is pointed out that in addition to its value in preventing miners' phthisis the influence of ventilation on efficiency is of great importance, and for that reason also ventilation of mines should receive most careful consideration.

Dr. Mavrogordato in his paper read last April points to experimental evidence from which it appears that good ventilation results in less dust.

Soluble Silica as the Cause of Miners' Phthisis.—A paper of special interest was contributed to this symposium in May, 1922, by Dr. Gye, of the National Institute for Medical Research, London. In this paper Dr. Gye puts forward the novel view, based on certain experiments performed by himself and his associates, that silicosis might be caused by soluble silica, i.e., the conversion in the tissues of SiO_2 into colloidal

silica. Dr. Gye's hypothesis—if proved—would mark a new and very important step forward in the study of miners' phthisis. At present one must acknowledge that the experimental evidence put forward by Dr. Gye is not entirely convincing.

Lectures to Miners.—A very interesting suggestion, and one which might receive further consideration by the Society, was made by Mr. Hendrikz, the late Organising Secretary of the Miners' Union, at the April, 1921, meeting. Mr. Hendrikz, who was himself a working miner and whose knowledge of the psychology of local miners is probably both extensive and reliable, stated that a large proportion of underground men are not conversant with the risks of miners' phthisis and the conditions which bring it about, and suggested that lectures might be given to working miners at the various mines by members of the Society.

Respirators.—At the same meeting Mr. E. M. Weston spoke of the possibility of using properly designed respirators as a means of preventing miners' phthisis. He stated that "gravimetric" tests showed an efficiency of about 95 per cent. at an experiment performed with a respirator invented by himself. I must point out, however, that "gravimetric" tests are not of very great value in ascertaining the efficiency of a respirator, because it is quite probable that a respirator which catches 95 per cent. by weight of dust would permit—in the 5 per cent. which passed through it—the passage of the most dangerous dust, the dust caught being made up of the large, heavy, but from a miners' phthisis point of view, harmless particles. From a practical point of view I am afraid respirators cannot be relied upon to play an important role in the prevention of miners' phthisis.

Improvements in Drills.—At the June, 1921, meeting, Mr. Alexander Richardson stated that a rotary type of drill would almost entirely eliminate dust, and that such a machine would be preferable commercially to anything at present on the market. It seemed to him that it was strange that the invention of a more suitable drill was not taken in hand by the

industry by means of systematic research and trial.

In the August number Mr. William Allen suggested the adaptation to the drill of a dust catcher, consisting of a soft cap fitting over the hole from which the dust would be extracted by a compressed air jet, and passed through a dust trap. I believe that experiments have for some time been, and are now being, carried out on these lines.

At the September, 1922, meeting Mr. Flugge-de Smidt reported the results of certain experiments carried out with the Leyner type of drill, modified with the idea of preventing excessive dust formation. The modifications made were based on the theory that the air which accompanies the water through the hollow steel is dust-laden as it emerges in the form of bubbles from the hole. Therefore various modifications were made in the drill to reduce the amount of air passing through the steel. The results obtained were on the whole satisfactory.

Atmospheric Conditions in Relation to Miners' Phthisis.—At the September, 1922, meeting Mr. J. D. Marquard read a very stimulating paper, in which he voiced his belief that the general atmospheric conditions in the mines have considerable bearing on the incidence of miners' phthisis. This paper was reviewed at some length by Mr. H. J. Ireland at the meeting of 17th February, 1923. Mr. Ireland agrees that the main contention of Mr. Marquard, which is in effect that hot, humid conditions act deleteriously on the lung tissues, is correct. He points out, however, that Mr. Marquard's suggestion that ventilation by compressed air is deleterious, because of the possible presence in the compressed air of fine oil particles, is not based on sound physiological data. I may add to this that I agree with Mr. Ireland. I would also reply in the negative to Mr. Marquard's question as to whether the presence of living organisms in the air is essential to health. It is only fair, however, to add to this that there does appear to be a substance in the atmosphere, the removal of which, by such means for instance as filtration through cotton wool, affects the "freshness" of the air, and makes it less pleasant. What this substance is, it is at present impossible to say. It was suggested that it is a catalyst. It is also possible that it is an electric charge.

Finally I will attempt to reply to some of the questions raised by Mr. Johnson at the January, 1923, meeting. Whether the

silica dust (SiO_2) is converted in the lungs into gelatinous silica is at present still under investigation.

So far as is known dumpsmen and people living in the vicinity of the dumps do not contract miners' phthisis—at least not in the advanced stages.

Silica is observable in the lungs *post mortem* as crystals of SiO_2 .

"Potters' Asthma" appears to be a very vague term. It probably also covers silicosis.

Mr. Johnson's suggestion that if there were no conversion of silica in the lungs the larger particles would do more harm than the minute particles, is one of the contentions of those who believe in the theory advanced by Dr. Gye.

I think, gentlemen, this is all I could cull from the published discussions of these meetings. I have put the synopsis as briefly as possible and without in any way trying to arouse a controversy, because all these points were fully discussed at the various meetings. I think the success of this Symposium has been such that the Society is justified in keeping in mind the possibility of having symposia on other subjects of general interest to our members, and I hope that the incoming Council will take this into consideration.

RECOVERY OF GOLD BY BLANKET CONCENTRATION IN SUBSTITUTION OF PLATE AMALGAMATION.

By F. WARTENWEILER.

(Printed in Journal, February, 1923.)

DISCUSSION.

Mr. Chas. E. Meyer: On reading Mr. Wartenweiler's valuable paper on the recovery of gold by blanket concentration in place of amalgamation, there appears to be one point which does not altogether conform with past experience. In most cases, where attempts have been made to recover gold from mine dumps the means adopted has been concentration. This seems to point to the fact that the values left in the dumps (though small) are closely associated with the pyritic portion of the ore, and were not

amenable to the treatment to which they were subjected.

Consequently if this concentration were removed it seems to follow that the cyanide charges, as well as the residues, would be lower in value, from the fact that the refractory portion of the ore had been eliminated prior to cyaniding.

Having been associated quite recently with a syndicate attempting to recover these values from old mine dumps, some of my experimental figures may prove of value to members.

Samples were taken by boring from the old concentrate dump at the Princess Estate and were tested in an ordinary elutriation apparatus consisting of four glass cones fed from a glass jar containing 10 assay tons of sample which had been neutralised and mixed with water. The U-neck of the first cone was filled with mercury, free from gold and silver; water under pressure was introduced into the jar containing the sample, and the pulp was forced through the apparatus having first to pass through the mercury before entering the cones. The following results were obtained:—

Assay value of original sample from dump, 3.09 dwt. per ton.

Extraction by amalgamation (from assay of mercury after experiment), 1.85 dwt. per ton=59.9 per cent.

After crushing the sample to —90, the same experiment yielded 2.329 dwt. per ton extraction by amalgamation=75.4 per cent.; and after crushing to —200 the extraction by amalgamation was found to be 2.527 dwt. per ton=81.8 per cent.

In all three cases quoted above perfect amalgamation was presumed, as contact of the ore and mercury was efficient, and the mercury did not sicken.

A proportionate quantity of the water used was assayed and found to contain dissolved gold equivalent to 0.509 dwt. per ton of sand.

From the above data the following results were deduced:—

Original Sample: Dissolved gold, 16.5 per cent.; free gold, 59.9 per cent.; encased gold, 23.6 per cent.

After crushing to —90: Dissolved gold, 16.5 per cent.; free gold, 75.4 per cent.; encased gold, 8.1 per cent.

After crushing to —200: Dissolved gold, 16.5 per cent.; free gold, 81.8 per cent.; encased gold, 1.7 per cent.

The dump from which samples were taken

had been lying subjected to weathering for many years.

The samples having been taken from boreholes extending from top to bottom of the dump, and since weathering would be very incomplete in depth it would seem that the amount of encased gold in the samples is low.

The ore was further tested for sulphur, and found to contain—

Sulphur =0.453 per cent.

Pyrites (Fe S_2) ... =0.849 per cent.

The figure for improvement in amalgamation results is further borne out by the author's results of experiments on concentrate (Nos. 4 and 7, p. 155).

This seems to me to be due to the removal of the encased gold with the free gold in the concentrate prior to treatment.

NOTES ON FAN VENTILATION IN DEEP MINES.

By J. A. WOODBURN.

(Printed in Journal, May, 1923.)

DISCUSSION.

Mr. A. J. Walton: Mr. Woodburn makes use of the generally accepted formula $p.a. = K S V^2$ to determine the ventilating pressure required for any given conditions. He states that the value of K. varies according to different authorities from .005lbs. per sq. foot to .02lbs. per sq. foot. In the Colliery Managers' Pocket Book for 1909, the following values are assigned to K.:—

Uniform pits and galleries: .00208lbs. per sq. foot.

Furnace pits: .00364lbs. per sq. foot.

Working places: .014lbs. per sq. foot.

Average of all galleries: .00936lbs. per sq. foot.

It is obvious from these figures that great care must be taken to select the value for K. which will give a correct result if the purchase of a fan is dependent upon the answer. There can be no doubt that the formula is correct, but it is quite impossible to calculate the rubbing surface of a stope

supported on timber packs, and almost equally impossible to ascertain the rubbing surface of any mine drive or cross cut.

Some years ago, whilst examining the ventilation problems of various mines on the Reef, I endeavoured to discover a value of *K*. for rectangular shafts. Fortunately at that time a large Sirocco fan was working in the Village Deep Mine, drawing air down both of the Village Deep shafts, and exhausting through the Village Main Mine. This fan was circulating approximately 272,000 cubic feet of air per minute, at a water gauge of 2.25 in., and by means of a sensitive barometer it was ascertained that 1.725 in. was due to friction in the Village Deep Mine, and that .525 in. was due to friction in the Village Main Mine. By taking the water gauge in a ventilating door at the bottom of one of the downcast shafts the nett water gauge for that shaft was obtained, and from this information a value for *K*. was worked out.

Further readings were taken at the other downcast shaft, and finally an average figure of .0037 lbs. per sq. foot was arrived at, which, though not within the limits mentioned by Mr. Woodburn, proved extremely useful in practice. In obtaining this figure the pipe and ladder way was taken to have half the area of the winding compartments. The rubbing surface consisted of the sum of the perimeters of all the compartments, multiplied by the depth of the shaft.

The factor of natural ventilation which must have been considerable in the case of the Village Deep, was eliminated in the calculations, as the fan was practically on the same horizon as the level to which the measurement of the shaft was taken.

If a calculation is being made to determine the water gauge required for a given quantity of air with the idea of ordering a fan it is well to err on the side of a low water gauge, as the lower the water gauge the larger the equivalent orifice of the mine, and the orifice of the fan should not be less than twice the orifice of the mine.

If the inlet of the fan is too large, very little loss in efficiency will result, whilst if

the inlet of the fan is too small a considerable amount of power will be absorbed in passing the air through the fan, causing a great loss of efficiency. Ample margin should be allowed in the motor or engine required to drive the fan unless all the factors are known within very narrow limits.

On many mines the use of auxiliary fans is highly desirable, but in the example quoted by Mr. Woodburn he suggests the installation of fans which are not necessary. If, as he suggests, a fan is placed on the surface to work at a water gauge of 4 in., it will be unnecessarily large in diameter, and a smaller fan could equally well do the work of the three low resistance splits with one auxiliary fan for the high resistance split. On the other hand it is obvious that if no surface fan is used, underground fans of smaller dimensions than required in the example given could do the work. The total horse power required under either of these schemes would be no more than that stated, and the capital outlay would be considerably less.

It is impossible to lay down any hard and fast rule for the ventilation of development ends. It will, however, generally be found that if rock drills are working at the face of a drive it is extremely difficult to do more than exhaust the amount of air made by the machines, leaving that portion of the drive behind the ventilation pipe almost stagnant. The exhaust pipe, further, does not induce any current of air to travel to the working face, unless placed almost in direct contact with it, a most difficult, if not impossible condition to attain. On the other hand if the ventilating pipe be used to force air into the face, the current of air will carry for quite an appreciable distance from the end of the pipe to the persons employed in drilling, and it appears to be much more important that they should receive an adequate supply of fresh air than the persons who are not so much exposed to dust. The drive away from the face is ventilated by all the exhaust air from the machines, as well as all the air which has entered through the ventilating pipe.

Mr. R. A. H. Flugge-de Smidt: The accompanying graphs show anemometer readings plotted against air pressures, and are the results of an attempt to compare the efficiency of a Venturi Booster and an Injector Blower against a jet in plain parallel piping.

Each type of blower was attached successively to approximately 60ft. of 1lin. diameter galvanised iron piping.

The identical jets were used for each type and the anemometer wired into a fixed position for all readings.

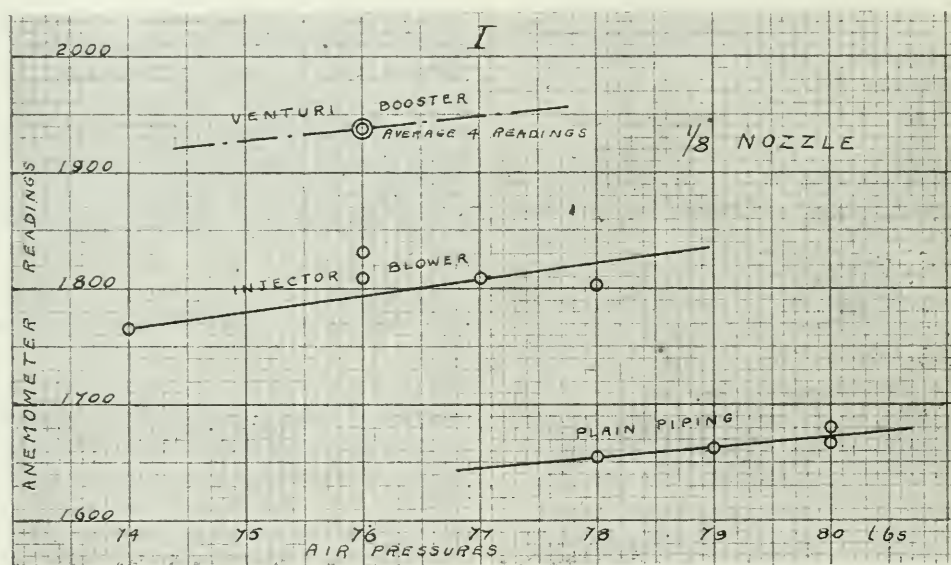
A stop-watch was used for timing.

It is interesting to note that where the air-pressures varied the anemometer readings responded on the whole extraordinarily

Watermeyer had been indefatigable in his efforts for the benefit of the Society, and he thought it was only right that they should show their very highest appreciation. In conclusion, he hoped that he would be able to fill the office as well as Professor Watermeyer had done it. (Applause.)

The President thanked members very much for the way in which they had received Mr. Watson's suggestion. He had only done his duty to the Society, and he felt very pleased that members thought it a success. (Applause.)

The meeting terminated at 9.55 p.m.



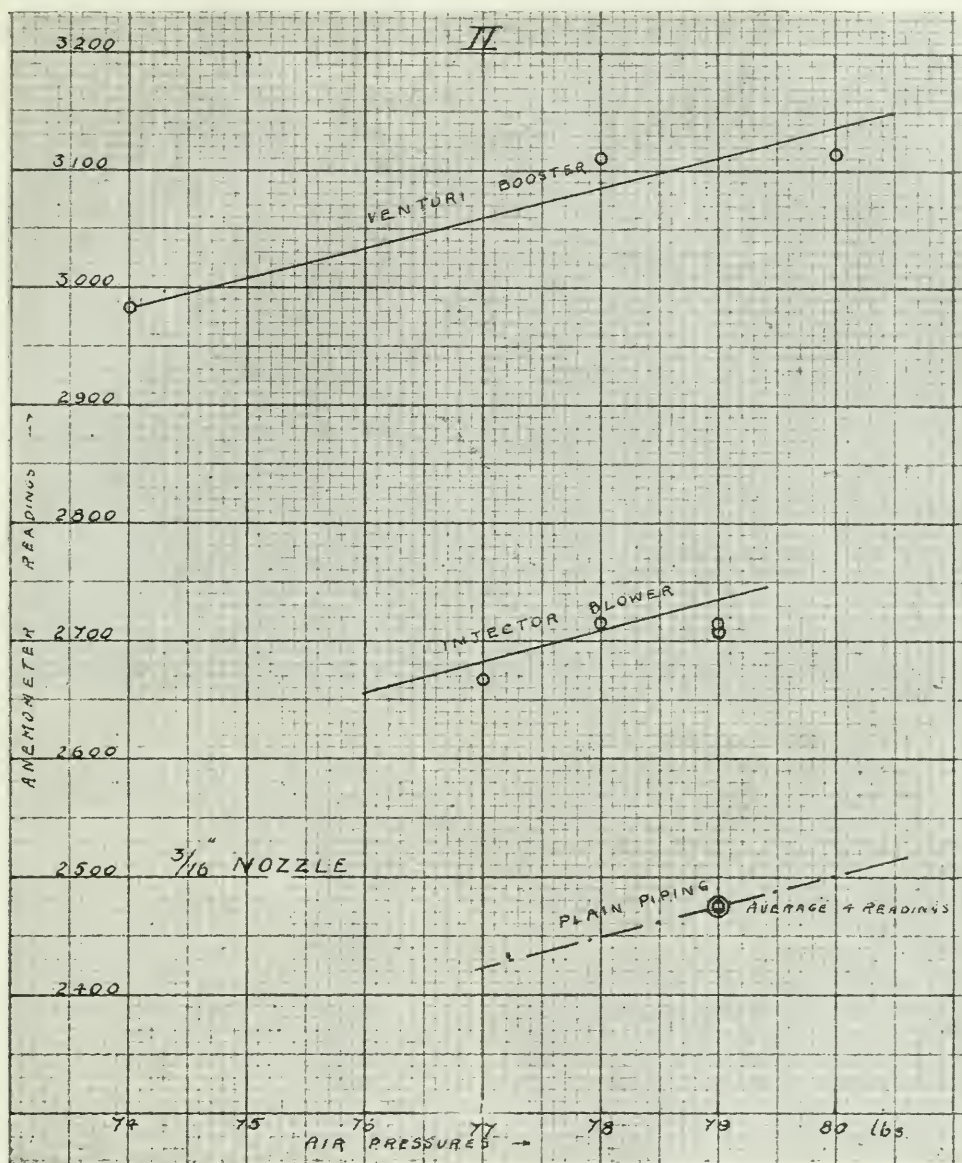
well considering the velocities recorded, also the larger the aperture of the nozzle the steeper the curve obtained.

Mr. F. W. Watson said there was one matter he would like to mention before the meeting closed. It was the last meeting of the year, and he was sure members present would like to show their appreciation on behalf of themselves and of the other members of the Society who were absent for the excellent services rendered to the Society by their retiring President, Professor Watermeyer. Speaking as a Member of Council and a Vice-President, he knew Professor

BOOK REVIEWS.

TECHNICAL METHODS OF ORE ANALYSIS FOR CHEMISTS AND COLLEGES, by Alfred H. Low, Sc.D., Professor of Chemistry, Colorado School of Mines. Ninth edition, entirely rewritten and enlarged. New York: John Wiley & Sons. London: Chapman & Hall.

The ninth edition of this well-known text book is very similar to its predecessor, published in 1919, but it has been considerably enlarged by the addition of a preliminary part consisting of outlines of methods for students' use.



The system adopted by the author in this section may seem unnecessarily detailed, but, as is observed in the remarks, many junior students seem unable to follow directions and obtain a reasonably accurate result.

The method of instruction used by the author is somewhat involved, but, in all probability, will be the quickest in the end as the chance of obtaining an inaccurate result is reduced to a minimum.

The technical methods of analysis are explained very fully, and a new departure consists in the addition of notes, expressly

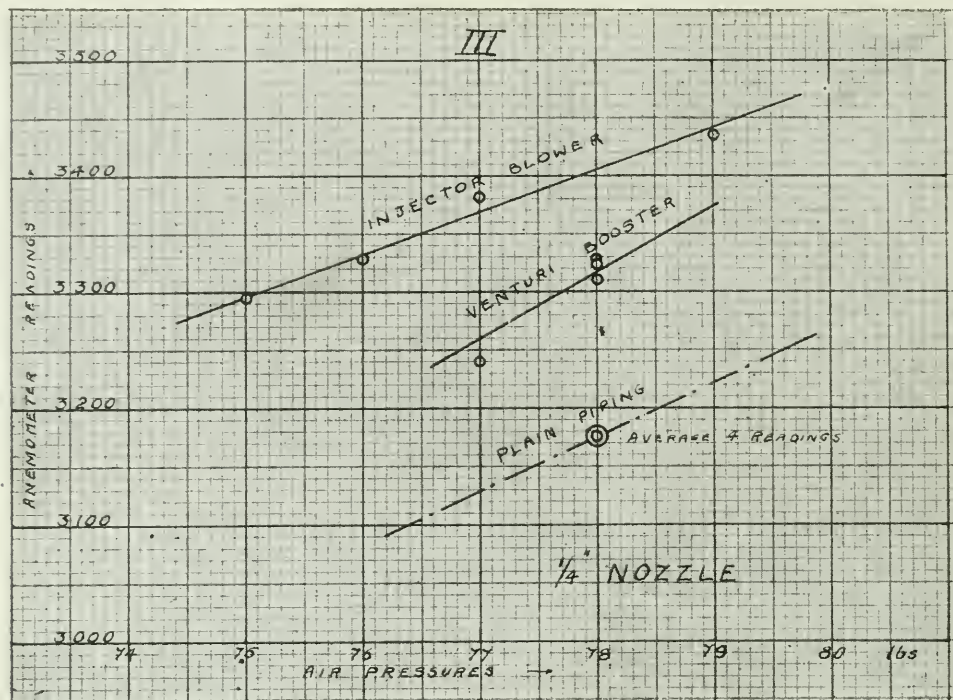
intended for students, which are to be found at the conclusion of many of the methods described.

The book is produced in the excellent style usually associated with the above firms' publications.

F.W.W.

TITANIUM, by A. H. A. Robinson. Canada Department of Mines, No. 579, 1922.

This publication, which has been briefly referred to in an abstract appearing in the *June Journal*, is a comprehensive summary



of the known Canadian deposits of titanium ores; of the physical and chemical properties of the element; the geological occurrence, nature and distribution of its ores; and the sources, production and uses of titanium.

The list of references to authorities and the bibliography of the literature of titanium are valuable additions to this work.

Details of attempts to concentrate the iron or the titanium rich portions of titaniferous magnetites show that, generally, very little success has been attained, though the results are of considerable scientific interest. The successful use of these magnetites in blast-furnaces appears to be proved, but there is no useful object attained by their employment when any titanium free ores are available, as the ores generally are of rather low iron content and the pig iron produced contains practically no titanium.

The two principal uses of titanium at present are as a deoxidiser of molten metals, for which purpose various titanium alloys are employed, and as a colouring material in the paint and ceramic industries, and also in the textile and leather industries in the form of dyes and mordants. It appears that South Africa might well consider the utilisation of her large titaniferous ore deposits for some of the purposes outlined.

An interesting process for the production of iron-free titanous acid with certain valuable by-products is described. This process—the Paquet-Andrien process—is carried out in three stages:—

1. Carbonitride of titanium and sodium cyanide are produced in a revolving furnace by heating titanium ore mixed with carbon and sodium carbonate in a current of nitrogen. In the presence of titanium ore it is claimed that the reaction can be brought about at a temperature below $1,000^{\circ}$.
2. The mixture is next subjected to the action of steam; titanate of soda, ammonia, and hydrocyanic acid being formed. The gases are absorbed, the ammonia in sulphuric acid, and the hydrocyanic acid in a soda solution.
3. The remaining mass is exhausted with hot water and the resulting solution of titanate of soda is decomposed by a current of carbonic acid.

By the Paquet-Andrien process there is thus obtained, beside titanous acid, ammonium sulphate and sodium cyanide as by-products and sodium carbonate is regenerated. Theoretically, 735 kg. of sodium cyanide will be produced for each metric ton of titanous acid.

R.A.C.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

BOOK REVIEW.—THE ANALYSIS OF COAL AND ITS BY-PRODUCTS. By S. ROY ILLINGWORTH, assisted by Jenkyn Griffiths, London. The Colliery Guardian Co., Ltd., 30-31, Fumival Street, Holborn, E.C. 4. (Price 21s.)—The author in the course of his professional activity has come to the conclusion that there exists the need for a book specially dealing with the analytical work required in the training of fuel and colliery chemists. Empiricism has, he thinks, characterised certain phases of the commercial analysis of coal and its products, with the consequence that the results obtained by buyer and seller have become a contentious matter, and the sale of coal on specification has made no headway in this country. Hence many people prefer to rely on the reputation of a coal which is more often than not the name of a colliery or a particular working, without being able to define the particular product or to check its delivery. The author has therefore endeavoured to review and scrutinise alternative methods of analysis in the hope that his book will not only be of value in the training of students, but will help towards the standardisation of commercial practice. At the outset he justly lays great stress on the correct sampling of coal, and proves by figures that unless a considerable bulk of coal is included in the sample, errors due to the "personal equation" of the sampler are more than likely to arise and that an agreement as to results between buyer and seller is practically impossible when the bulk sample collected is less than, say, 600 lb. The U.S. Bureau of Mines has been conducting a series of investigations, and published several bulletins on the question of sampling coal, and the salient features of their method are reproduced in the present book. The author next proceeds to deal with the proximate and ultimate analysis of coal, the determination of moisture, fixed carbon, volatile matter, ash, sulphur, nitrogen, etc., in coal, devoting special chapters to the ash in coal and to coking coals and coke. Further chapters deal with the analysis of crude coal tar, the examination of light oil and its products, the analysis of the heavier fractions of coal tar and products derived therefrom, the analysis of ammonia liquor, ammonium salts and spent oxide and with gas analysis, including the analysis of "mine" air. The chapters next following are devoted to calorimetry of solid, liquid and gaseous fuels, to control of combustion and the action of solvents on coal. Factors influencing the flashpoint and the Redwood viscometer are (as applied to oils) dealt with next. Finally, in five appendices are dealt with: (1) Spontaneous combustion of coal; (2) analysis of coal and stone dust samples; (3) the analysis of sulphuric acid; and (4) of caustic soda; while Appendix V. contains various tables and constants. There are 63 illustrations in the text, and two folding plates of six graphs by Drs. Colman and Yeoman relating to various by-products of tar. The methods described in the book are the latest. Where older

methods are given they are accompanied by useful criticism in justification of the adoption of more precise processes, many of these being modifications suggested by the author's personal researches. It is hoped that the volume will be followed by another giving details of actual results. Many enquiries reach this country from abroad as to particulars of the better known coals in the various fields. Such a collection of data was published many years ago, but the book has long been out of print, the coals may have changed in the different seams, and, of course, methods of sampling and analysis have improved.—*Iron and Coal Trades Review*, October 13, 1922, p. 539. (J.A.W.)

METHODS OF ANALYSIS OF COMMERCIAL SOAPS AND SOAP PRODUCTS.—A Committee of the American Chemical Society has investigated the methods in general use for the analysis of commercial soaps and soap products, and issued a report embodying their conclusions. This report has lately been revised because the Committee has attempted to cover methods that would be of importance in commercial transactions rather than methods that would be mainly of theoretical interest and only applicable in special cases.

The first part of the report concerns methods of sampling and then the preparation of the sample for analysis is dealt with. The analytical section itself includes the estimation of matter volatile at 105 degrees C.; the total matter insoluble in alcohol; the combined alkali and total anhydrous soap; chlorides; unsaponified and unsaponifiable matter; rosin; fatty acids; borax; carbonates; phosphates; silicates; sugar and starch; glycerol, etc.

A fuller report on the subject will be found in the December, 1922, issue of the *Journal of Industrial and Engineering Chemistry*.—COMMITTEE, AMERICAN CHEMICAL SOCIETY. *The Chemical Trade Journal* (also *Journal Indust. Engrin. Chemistry*), December 29, 1922, p. 793. (R.A.C.)

NEW COMPOUNDS OF PLATINUM, PALLADIUM, IRIUM AND RHODIUM AND A NEW METHOD FOR THEIR ESTIMATION.—When 0.25 per cent. solutions of potassium chloroplatinite and sodium thiocyanate are mixed and left for five or six hours at the ordinary temperature, then shaken with 50 gm. of ammonium nitrate or sulphate per litre, a light brown, flocculent precipitate of platinum thiocyanate is produced.

The precipitate is soluble in aqua-regia and in potassium cyanide solution, but not in single acids or alkalis. If the precipitation is carried out in boiling solutions a black flocculent precipitate having the composition $Pt_4(OH)_4\dot{S}(CS.NH_2)_4$ is obtained. On ignition, a residue of platinum is left so that the compound is suitable for use in the estimation of platinum. Under similar conditions palladium gives a precipitate which, on ignition, gives a fusible residue of Pd_3S which is slowly transformed, after long heating, into the metal. Rhodium also gives a black precipitate which, on ignition, yields metallic rhodium. Iridium is incompletely precipitated as a flocculent, yellow compound.—W. N. IVANOV, *Journal Chemical Society (Chem. Zeitung)*, 1923, 47: 209-210. April, 1923, p. ii., 263. (R.A.C.)

IODINE PENTOXIDE CARBON-MONOXIDE INDICATOR.—In the *Journal of Industrial and Engineering Chemistry*, Messrs. S. H. Katz and J. J. Bloomfield, of the Pittsburgh Experiment Station, Bureau of Mines, U.S.A., give details of tests carried out with a carbon-monoxide indicator, referred to as the Hoolamite or activated iodine pentoxide indicator. The indicator gives positive indications for carbon monoxide in air in concentrations of 0.07 per cent., or more. In a few instances there were no indications with 0.07 and 0.08 per cent. No failures to indicate occurred at higher concentrations. Two tests with 0.10 per cent. CO in air indicated 0.10 and 0.05 per cent.; eight tests with 0.15 per cent. gas gave determinations ranging from 0.10 to 0.23 per cent., with an average of 0.16. At higher concentrations the variations were proportionately about the same. Determinations of carbon monoxide are easily made in less than one minute, and no skill is required. Only one person in thirty has been found to have difficulty in determining the colour shades. The instrument is proving a valuable aid in mine-rescue and recovery operations after fires and explosions, and for testing air around blast furnaces, gas producers, water-gas and coal-gas plants, and other places where carbon monoxide occurs. It also may be used to promote furnace economy by indicating carbon monoxide in flue gases.—S. H. KATZ and J. J. BLOOMFIELD, *Iron and Coal Trades Review*, March 9, 1923, p. 353. (J.A.W.)

MISCELLANEOUS.

RESUSCITATION FROM ELECTRIC SHOCK.—*The McLaglan System.*—Some time ago the author's attention was drawn to a new system of resuscitation, and it appeared to him then to be a very practicable method in cases of electric shock. The system was certainly not new, because it had been known as far back as 200 years B.C. It was only in recent years, however, that it had come to the knowledge of a Britisher. It had now been termed the McLaglan system of resuscitation, after the name of the individual who was fortunate enough to discover the secret from the Japanese. It was based on the ancient "katsu" system—the word "katsu" meaning resuscitation.

The author went on to explain that he was indebted to Captain Leopold McLaglan for the necessary permission to bring the system before the National Association of Colliery Managers and the Association of Mining Electrical Engineers. The system operated by pressure on certain nerve centres which stimulated the nerves and caused a reaction of the heart. The nerves which were affected were those of the pectoral arch, where the thinness of the tissue and other matter left the nerves to the heart most exposed. The other nerve centres affected were the pneumogastric centre and the auditory nerves to the brain. As the nerves were still alive after the heart's action had ceased due to electric shock, a person was not actually dead until decomposition or putrefaction set in—the time for this to take place depending, of course, on the cause of the stoppage of the heart's action and also on the condition of health

of the person involved. With the aid of the following instructions it was quite easy to apply the McLaglan system of resuscitation.

Place the subject in a sitting position, supported by the operator's right or left knee. Place the thumbs at the pectoral arch where the muscles leave the nerves most exposed. Apply pressure at this point with the fingers resting firmly on the diaphragm of the chest, the pressure being applied with both fingers and thumbs downwards and upwards. This must be continued for some time, and the effect was to stimulate the nerves of the heart which caused the diaphragm to slightly expand and contract, thereby making the subject inhale and exhale. Just as the thumbs were applying the pressure in their upwards and downwards motion on the chest, a blow was delivered by the operator's knee on the seventh dorsal vertebrae. This tended to stimulate the pneumogastric centres and assisted in causing the diaphragm to expand and contract. Greater inhalation and exhalation in this way brought the dilation of the heart to its fullest capacity, and the blood was circulated through the body. Another point where the blow could be delivered was the third from the last dorsal vertebrae, which tended to stimulate the nerves of the spine.

Electric shock undoubtedly affected the whole nervous system, and a state of unconsciousness—and even death—were brought about from this cause. The McLaglan system appeared to be the only practical one for bringing about resuscitation, because it primarily dealt with the expansion and contraction of the diaphragm. At the same time the thumb pressure at the pectoral arch, together with the blow which was delivered on or about the seventh dorsal vertebrae, and shouting as the blow was delivered, tended to stimulate the auditory nerves of the brain, and acted as a counter-shock, thereby bringing about a condition of resuscitation.

There seemed to be no doubt that the McLaglan system of resuscitation showed marked advantage over the Sylvester, Schäfer, and Labord, because it was applied to the life-giving centres of the human body rather than to the muscles and arms only.—JOHN A. BROWN, *Iron and Coal Trades Review*, March 2, 1923, p. 313. (J.A.W.)

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- 1905 MAIN, A. F., P.O. Box 109, -Bis, Mexico City, Mexico.
- 1916 MANN, J. D., New Unified Main Reef G.M. Co., Ltd., P.O. Box 5, Maraisburg, Transvaal.
- 1910 MARCERISON, S. B., Totok (Noord-Celebes), D.E. Indies.
- 1911 MARKS, L., P.O. Box 196, Germiston, Transvaal.
- 1902 MARQUARD, J. D., Mines Dept., P.O. Box 1072, Johannesburg.
- 1898 MARRIOTT, H. F. (*Life Member*), "Heids," Morland Close, Hampstead Way, London, N.W.11.
- 1914 MARTIN, H. Rose, Cullinan Buildings, P.O. Box 5221, Johannesburg.
- 1908 MARTIN, H. Stuart, Rand Mines, Ltd., P.O. Box 1056, Johannesburg.
- 1921 MARTIN, W., African Iron and Steel Products, Ltd., Main Reef Road, Denver, Johannesburg.
- 1911 MARTYN, T. Graham, New Unified Main Reef G.M. Co., Ltd., P.O. Box 5, Maraisburg, Transvaal.
- 1921 MASKEW, F. W., Brakpan Mines, Ltd., P.O. Box 3, Brakpan, Transvaal.
- 1897 MATTHEWS, E. L., New Modderfontein G.M. Co., Ltd., P.O. Box 25, Benoni, Transvaal.
- 1910 MCPHEE, J., Transvaal and Delagoa Bay Colliery, Witbank, Transvaal.
- 1916 MEIKLEJOHN, C. A., Rand Mines, Ltd., P.O. Box 1056, Johannesburg.
- 1918 MELLOR, E. T., Central Mining and Investment Corporation, Ltd., P.O. Box 1169, Johannesburg.
- 1895 MELVILL, G., Nourse Mines, Ltd., P.O. Box 32, Denver, Johannesburg.
- 1904 MEYER, C. E., P.O. Box 101, Crown Mines, Johannesburg.
- 1910 MEYER, E. C. J., East Rand Prop. Mines, Ltd., P.O. Box 57, East Rand, Transvaal.
- 1906 MILLS, F., Langlaagte Estate and G.M. Co., Ltd., P.O. Box 98, Langlaagte, Transvaal.
- 1912 MILLS, L. D., 121, Second Street, San Francisco, Cal., U.S.A.
- 1917 MITKE, C. A., P.O. Box 2051, Bisbee, Arizona, U.S.A.
- 1902 MOIR, J., Govt. Chemical Laboratories, P.O. Box 1080, Johannesburg.
- 1918 MORGAN, P. G., 50, Marjoribanks Street, Wellington, New Zealand.
- 1897 MORRELL, H. H., Geldenhuis Deep, Ltd., P.O. Box 54, Cleveland, Johannesburg.

- 1896 MORRISBY, P. T. B., Witwatersrand G.M. Co., Ltd., P.O. Box 1, Knights, Transvaal.
- 1908 MOSSOP, W. C., Wolluter Gold Mines, Ltd., P.O. Box 1160, Johannesburg.
- 1909 NEAL, W., Hotel Cosmopolita, Guadalajara, Jalisco, Mexico.
- 1913 NEILL, J. M., Slaaihoek Mines, Ltd., P.O. Airlie, Transvaal.
- 1920 NESS, T. R. (Address wanted).
- 1922 NEUFELD, E., New Prospect Soap and Chemical Manfg. Co., Ltd., P.O. Box 73, Denver, Johannesburg.
- 1903 NEWHALL, P. M., Central Mining and Investment Corp'n., Ltd., P.O. Box 1169, Johannesburg.
- 1902 NEWTON, S., Modderfontein "B" Gold Mines, Ltd., P.O. Modderbee, Transvaal.
- 1902 NIVEN, R., P.O. Box 4022, Johannesburg.
- 1911 NOBLE, E., Lehong Tandai, Res. Benkoelen, Sumatra, D.E. Indies.
- 1904 NORMAND, J. N., Transvaal G.M. Estates, Ltd., Pilgrims Rest, Transvaal.
- 1899 NORTH, W. J. R., Modderfontein Deep Levels, Ltd., P.O. Box 326, Benoni, Transvaal.
- 1897 O'HARA, J. D., Geldenbuis Deep, Ltd., P.O. Box 54, Cleveland, Johannesburg.
- 1910 OLSEN, A. C. L., Modderfontein East, Ltd., P.O. Box 14, Springs, Transvaal.
- 1916 ORENSTEIN, A. J., Rand Mines, Ltd., P.O. Box 1056, Johannesburg.
- 1908 ORR, J., University of the Witwatersrand, P.O. Box 1176, Johannesburg.
- 1920 ORTLEPP, J. A., P.O. Frederikstadt, Transvaal.
- 1898 OSBORN, S. S., Glynn's Lydenburg, Ltd., P.O. Box 23, Sabie, Transvaal.
- 1920 PAINE, H. H., University of the Witwatersrand, P.O. Box 1176, Johannesburg.
- 1905 PAM, E., Modderfontein East, Ltd., P.O. Box 14, Springs, Transvaal.
- 1920 PARRY, J., The Laboratory, De Beers Consolidated Mines, Ltd., Kimberley, C.P.
- 1907 PATTERSON, P. F., Falcon Mines, Ltd., Umvuma, S. Rhodesia.
- 1903 PAYNE, A. E., Cons. Langlaagte Mines, Ltd., P.O. Box 15, Langlaagte, Transvaal.
- 1916 PEARCE, J. W., Nil Desperandum Mine, P.O. Shabani, S. Rhodesia.
- 1894 PEARCE, S. H. (*Hon. Life Member*), P.O. Box 102, Crown Mines, Johannesburg.
- 1909 PHILLIPS, F. D., Robinson Deep, Ltd., P.O. Box 1392, Johannesburg.
- 1922 PHILLIPS, F. N., Central Mining and Investment Corp'n., P.O. Box 149, Johannesburg.
- 1908 PHILLIPS, J. C., Govt. G.M. Areas, Ltd., P.O. Box 12, State Mines, Transvaal.
- 1919 PIROW, H., Mines Dept., P.O. Box 1072, Johannesburg.
- 1910 PITT, C. K., Brakpan Mines, Ltd., P.O. Box 3, Brakpan, Transvaal.
- 1922 POOLE, S. E., Witwatersrand G.M. Co., Ltd., P.O. Box 1, Knights, Transvaal.
- 1908 POPHAM, J. L., Mining and Metallurgical Club, 3, London Wall, London, E.C.
- 1921 POWER, R. C., P.O. Box 5650, Johannesburg.
- 1920 PRENTICE, T. K., Knight Central, Ltd., P.O. Box 91, Germiston, Transvaal.
- 1920 PROBERTS, T., P.O. Box 1091, Johannesburg.
- 1897 PROUT, W. M., Leonie, Pienaar's River, Dist. Pretoria.
- 1909 PRYCE, L., Central Mining and Investment Corp'n., Ltd., P.O. Box 4563, Johannesburg.
- 1919 PULFORD, A. E., Ferreira Deep, Ltd., P.O. Box 1021, Johannesburg.
- 1902 PYLES, J. F., Village Deep, Ltd., P.O. Box 1064, Johannesburg.

- 1920 RAMSAY, R., Princess Estate and G.M. Co., Ltd., P.O. Box 112, Roodepoort, Transvaal.
- 1920 RAY, L. G., Mines Dept., P.O. Box 405, Krugersdorp, Transvaal.
- 1917 RAYMOND, R. M., 61, Broadway, New York City, U.S.A.
- 1919 READ, H. A., P.O. Box 6403, Johannesburg.
- 1910 REDFERN, A., Grenfell Colliery, Ltd., P.O. Box 110, Ermelo, Transvaal.
- 1911 REID, M. H., Witwatersrand Deep, Ltd., P.O. Box 5, Knights, Transvaal.
- 1921 REIM, E. P., Technical College, Durban, Natal.
- 1920 RICHARDSON, J., Witwatersrand G.M. Co., Ltd., P.O. Box 1, Knights, Transvaal.
- 1922 RIDGE, H. M., 2, Gt. Winchester Street, London, E.C.2.
- 1921 RIGGS, C. H., Harding Street, Newcastle, Natal.
- 1920 RITCHIE, J., Transvaal Silver and Base Metals, Ltd., P.O. Argent, Transvaal.
- 1912 ROBERTS, E. J. (Address wanted).
- 1910 ROBERTS, F. G. A., P.O. Box 809, Johannesburg.
- 1920 ROBERTSON, L. A., c/o Witbank Colliery, Ltd., Witbank, Transvaal.
- 1917 ROGERS, A. W., P.O. Box 401, Pretoria.
- 1912 ROME, A. S., The Cam and Motor G.M. Co., Ltd., Eiffel Flats, Rhodesia.
- 1922 ROSS, G. C., 35, Nelson Road, Booyens, Johannesburg.
- 1921 RUNCIMAN, A. W., P.O. Box 103, Crown Mines, Johannesburg.
- 1903 RUSSELL, S., P.O. Box 393, Benoni, Transvaal.
- 1921 SCHLESINGER, B., P.O. Box 7448, Johannesburg.
- 1913 SCRYMGEOUR, A., P.O. Graskop, Transvaal.
- 1922 SEAGER, J. A., P.O. Box 6740, Johannesburg.
- 1917 SELBY, P., Ferreira Deep, Ltd., P.O. Box 1021, Johannesburg.
- 1895 SHALLCROSS, V. F. (*Life Member*), St. George's House, 117, St. George's Terrace, Perth, Australia.
- 1909 SHARWOOD, W. J., Homestake Mining Co., Lead, S. Dakota, U.S.A.
- 1909 SIMPSON, C. B., Geldenhuis Deep, Ltd., P.O. Box 54, Cleveland, Johannesburg.
- 1921 SIMPSON, T. R., Minerals Separation, Ltd., P.O. Box 2959, Johannesburg.
- 1922 SMIDT, R. A. H. Flugge-de, Modderfontein Deep Levels, Ltd., P.O. Box 326, Benoni, Transvaal.
- 1920 SMIDT, S. de, Mines Dept., P.O. Box 405, Krugersdorp, Transvaal.
- 1910 SMITH, G. Hildick, Robinson G.M. Co., Ltd., P.O. Box 1024, Johannesburg.
- 1921 SMITH, H. Hardy, Minerals Separation, Ltd., P.O. Box 2959, Johannesburg.
- 1921 SMITS, B. A. S., P.O. Box 858, Johannesburg.
- 1905 SMYTHE, J. J. R., Ferreira Deep, Ltd., P.O. Box 1021, Johannesburg.
- 1921 SOAR, E., P.O. Box 1, Witbank, Transvaal.
- 1909 SOLOMON, L., P.O. Box 40, Davel, District Ermelo, Transvaal.
- 1898 SÖRENSEN, S. S., Braden Copper Co., 120, Broadway, New York, U.S.A.
- 1903 SPOOR, A. L., P.O. Box 7312, Johannesburg.
- 1905 STANLEY, G. H., University of the Witwatersrand, P.O. Box 1176, Johannesburg.
- 1903 STEELS, S. H., Geldenhuis Deep, Ltd., P.O. Box 54, Cleveland, Johannesburg.
- 1910 STEVENS, T. B., P.O. Box 5, Fimiston, W. Australia.
- 1904 STOKES, R. S. G., De Beers Consolidated Mines, Ltd., Kimberley, C.P.
- 1919 SWINBURNE, U. P., Mines Department, P.O. Box 1072, Johannesburg.
- 1907 SYDENHAM, C. T., Luipaardsvlei Estate and G.M. Co., Ltd., P.O. Box 53, Krugersdorp, Transvaal.

- 1921 SYMONS, C. J. D., P.O. Box 469, Johannesburg.
- 1897 TAVENER, P. S., P.O. Koosfontein, via Bloemhof, Transvaal.
- 1909 TAYLER, W. A. C., Simmer and Jack G.M. Co., Ltd., P.O. Box 192, Germiston, Transvaal.
- 1902 TAYLOR, J. A., Simmer and Jack G.M. Co., Ltd., P.O. Box 192, Germiston, Transvaal.
- 1920 TELFORD, W., Village Deep, Ltd., P.O. Box 1064, Johannesburg.
- 1908 THOMAS, A., Simmer and Jack G.M. Co., Ltd., P.O. Box 192, Germiston, Transvaal.
- 1904 THOMAS, F. J., Eileen Alannah Mine, Eiffel Flats, S. Rhodesia.
- 1898 THOMAS, J. E., P.O. Kraal, Transvaal.
- 1917 THOMSON, A. R., Wankie Colliery Co., Ltd., Wankie, Rhodesia.
- 1909 THORBURN, J. M., P.O. Box 1214, Johannesburg.
- 1917 THORLUND, J., P.O. Box 4363, Johannesburg.
- 1917 THORP, M. J., Cons. Main Reef Mines and Estate, Ltd., P.O. Box 2, Maraisburg, Transvaal.
- 1910 THORPE, W. E., Geduld Proprietary Mines, Ltd., P.O. Dersley, Transvaal.
- 1903 THURLOW, J. R., Rand Mines, Ltd., P.O. Box 1056, Johannesburg.
- 1903 THURSTON, G. H., 3, Old Jewry, London, E.C.2.
- 1910 TILLARD, M. O., New Modderfontein G.M. Co., Ltd., P.O. Box 25, Benoni, Transvaal.
- 1918 TODD, R., P.O. Box 2762, Johannesburg.
- 1909 TOOMBS, C., P.O. Box 149, East Rand, Transvaal.
- 1919 TRAVERS, J. W., Knight Central, Ltd., P.O. Box 91, Germiston, Transvaal.
- 1919 TREDGOLD, R. H., Witwatersrand G.M. Co., Ltd., P.O. Box 1, Knights, Transvaal.
- 1910 TRELOAR, A., Tyndrum Lead and Zinc Mines, Ltd., Tyndrum, Perthshire, N.B.
- 1902 TREVOR, T. G., Mines Department, P.O. Box 401, Pretoria.
- 1910 TRIGGS, J. T., Nourse Mines, Ltd., P.O. Box 32, Denver, Johannesburg.
- 1904 TRUSCOTT, S. J. (*Life Member*), 32, Netherhall Gardens, London, N.W.
- 1921 TUDHOPE, C. D., City Deep, Ltd., P.O. Box 1411, Johannesburg.
- 1905 TURVEY, W. E., Robinson Deep, Ltd., P.O. Box 1392, Johannesburg.
- 1919 TUTT, C. J., S. Van Ryn Reef G.M. Co., Ltd., P.O. Box 41, Nigel, Transvaal.
- 1921 UNGER, F. A., Cons. Mines Selection Co., Ltd., P.O. Box 1048, Johannesburg.
- 1920 VANDERMAELEN, C. A., Union Miniere du Haut Katanga, Elisabethville, Belgian Congo.
- 1903 VAUGHAN, J. E., Mines Dept., P.O. Box 376, Pietermaritzburg, Natal.
- 1904 VEASEY, J. H., P.O. Box 1251, Johannesburg.
- 1921 VEITCH, N. A., Minerals Separation, Ltd., P.O. Box 2959, Johannesburg.
- 1920 WADDINGTON, S. C. (Address wanted).
- 1916 WAITES, T. P. (Address wanted).
- 1907 WALKER, J. F., Cons. Main Reef Mines and Estate, Ltd., P.O. Box 2, Maraisburg, Transvaal.
- 1922 WALLER, F. H. G., 160, Tramway Street, Kenilworth, Johannesburg.

- 1910 WALTON, A. J., P.O. Box 102, Crown Mines, Johannesburg.
1921 WARD, W. G., P.O. Box 668, Johannesburg.
1911 WARTENWEILER, F., Central Mining and Investment Corp., Ltd., P.O. Box 1169, Johannesburg.
1914 WATERMEYER, G. A., University of the Witwatersrand, P.O. Box 1176, Johannesburg.
1903 WATSON, F. W., Rand Mines, Ltd., P.O. Box 1056, Johannesburg.
1897 WATSON, J., P.O. Box 6401, Johannesburg.
1921 WEBBER, H. O'K., Luncarty, Claremont, C.P.
1910 WELLS, E. F. V., Tweefontein Colliery, P.O. Coalville, Transvaal.
1909 WESSELS, J. J., The Afrikander Lease, P.O. Box 121, Klerksdorp, Transvaal.
1908 WESTON, E. M., P.O. Box 3308, Johannesburg.
1897 WHITBY, A., P.O. Box 849, Johannesburg.
1897 WHITE, H. A., Geduld Proprietary Mines, Ltd., P.O. Dersley, Transvaal.
1902 WHYTEHEAD, E. H., New Transvaal Chemical Co., Ltd., P.O. Delmore, Transvaal.
1904 WILEY, H., Game Park Farm, Gwelo, S. Rhodesia.
1919 WILKES, H. R. S., United Party Club, P.O. Box 5352, Johannesburg.
1902 WILKINSON, J. A., University of the Witwatersrand, P.O. Box 1176, Johannesburg.
1919 WILLCOX, A., Witwatersrand G.M. Co., Ltd., P.O. Box 1, Knights, Transvaal.
1918 WILLCOX, J. S. (Address wanted).
1920 WILLEY, J. L., Cons. Mines Selection Co., Ltd., P.O. Box 2269, Johannesburg.
1905 WILLIAMS, A. F., De Beers Cons. Mines, Ltd., Kimberley, C.P.
1921 WILLIAMS, A. T., 41, Lonsdale Road, Gorse Hill, Stetford, Manchester, England.
1902 WILLIAMS, G. W., 36, Acacia Road, St. John's Wood, London, N.W.8.
1894 WILLIAMS, J. R. (*Hon. Life Member*), 61, Park Lane, Parktown, Johannesburg.
1909 WILLIAMS, W. O., Meyer & Charlton G.M. Co., Ltd., P.O. Box 27, Jeppes, Johannesburg.
1920 WILMOT, L. B., P.O. Box 2106, Johannesburg.
1911 WILSON, J. C., Messina, Zoutpansberg, Transvaal.
1920 WILSON, W. P., P.O. Box 1091, Johannesburg.
1920 WINCKWORTH, L., Modderfontein "B" G.M., Ltd., P.O. Modderbee, Transvaal.
1903 WINSLOW, G. (*Life Member*), Messrs. Patullo, Fords & Co., 65, London Wall, London, E.C.
1919 WONNACOTT, R. H., Miners' Training School, P.O. Box 82, Jeppes, Johannesburg.
1922 WOOD, A. A., Cons. Langlaagte Mines, Ltd., P.O. Box 15, Langlaagte, Transvaal.
1920 WOODBURN, J. A., P.O. Box 396, Johannesburg.
1899 WORMALD, R., Montrose Mine, P.O. Ivy, Barberton, Transvaal.
1920 WRAITH, C. O., Bwana M'Kubwa, N. Rhodesia.
1910 WYLY, A. J., Lebong Donok, Moeara Aman, Sumatra, D.E. Indies.

1896 YATES, J., Jan Smuts Avenue, North of Zoo, Johannesburg.

ASSOCIATES.

Year of
Admission.

- 1921 ALEXANDER, S., P.O. Box 1842, Johannesburg.
 1902 ANDERSON, A. G. (Address wanted).
 1917 ANDERSON, A. M., Rand Mutual Assurance Co., Ltd., P.O. Box 413, Johannesburg.
 1919 ANDERSON, R. C., New Modderfontein G.M. Co., Ltd., P.O. Van Ryn, Transvaal.
 1913 ANDERSON, W., P.O. Box 550, Johannesburg.
 1906 ANDREWS, A. A., Maid of De Kaap Mine, P.O. Noordkaap, Transvaal.
 1917 ANDREWS, E. T. E., Barberton, Transvaal.
 1913 AYMARD, M., c/o The Technical Bookshop, Salisbury House, London, E.C.
- 1911 BALL, H. Standish, Apex (Trinidad) Oilfields, Fyzabad, Trinidad, B.W. Indies.
 1910 BANKS, J. H. G., c/o Waihi G.M. Co., Ltd., Waikino, Auckland, New Zealand.
 1921 BARRY, E. A., Modderfontein "B" G.M., Ltd., P.O. Modderbee, Transvaal.
 1912 BEARDWOOD, J. P., P.O. Box 2028, Johannesburg.
 1911 BELL, H. C. F., P.O. Box 9, Pilgrims Rest, Transvaal.
 1905 BOLITHO, E. J., Modderfontein "B" School, P.O. Modderbee, Transvaal.
 1910 BRINSDEN, F. G. S., Kalgurli G.M. Co., Ltd., Fimiston, W. Australia.
 1914 BRISE, H. R. Ruggles, Rosehaugh Co., Dar-es-Salaam, Tanganyika Territory.
- 1907 CAMERON, L. D., 34, Prince Albert Road, Mosman, New South Wales.
 1909 CLARENCE, G. J. V., New Consolidated Goldfields, Ltd., P.O. Box 1167, Johannesburg.
 1909 CLEVINGER, G. H., 132, Clark Road, Brookline, Mass., U.S.A.
 1907 COGLE, C. T., Land Survey Dept., Nairobi, Kenya Colony, E. Africa.
 1897 COOK, J. F., Whitney Point, New York, U.S.A.
 1911 COWLES, E. P., Union Corporation, Ltd., P.O. Box 1156, Johannesburg.
 1918 CROXFORD, L. J., Witwatersrand G.M. Co., Ltd., P.O. Box 1, Knights, Transvaal.
- 1919 DALY, W. E., Falcon Mines, Ltd., Umyuma, S. Rhodesia.
 1913 DAVIDSON, E. G., c/o L. Hauck, P.O. Box 72, Caracas, EE. UU. de Venezuela.
 1899 DAVIDSON, J., Johannesburg Consolidated Investment Co., Ltd., P.O. Box 231, Johannesburg.
 1919 DAVISON, H., Morgan Crucible Co., Ltd., Battersea Works, London, S.W.
 1904 DEAKIN, C. E., Village Deep, Ltd., P.O. Box 1064, Johannesburg.
 1910 DEGENHARDT, W. R., 1, London Wall Buildings, London, E.C.2.
 1906 DENNEHY, M. J., Apex Reduction Works, P.O. Box 2, Benoni, Transvaal.
 1905 DEWAR, C. L., Van Ryn Deep, Ltd., P.O. Box 225, Benoni, Transvaal.
 1911 DICEY, E. C., Consort Mine, P.O. Noordkaap, Transvaal.
 1921 DIERING, J. H. A., Vaalhoek Mine, P.O. Box 105, Pilgrims Rest, Transvaal.
 1922 DONALD, J. E., P.O. Box 491, Johannesburg.
 1913 DONKIN, W., Shamva Mines, Ltd., Shamva, S. Rhodesia.

- 1905 DORR, J. V. N., 101, Park Avenue, New York City, U.S.A.
1908 DOWSETT, C. W., Dome Mines, S. Porcupine, Ontario, Canada.
1898 DURHAM, B. W., Lonely Reef G.M. Co., Ltd., Lonely Mine P.O., S. Rhodesia.
- 1919 EAMES, L. B., 1720, California Street, Denver, Colorado, U.S.A.
1912 EDGAR, J., P.O. Box 11, Pilgrims Rest, Transvaal.
1920 EDMANDS, H. R., Globe and Phoenix G.M. Co., Ltd., Que Que, S. Rhodesia.
1917 ELLIOTT, E. M., Union Minière du Haut Katanga, Likasi, Katanga, Belgian Congo.
1898 EVANS, S., 153, Nugget Street, Johannesburg.
1917 EXTER, J. K., 159, Rosetta Street, Pretoria, W.
- 1920 FERRERO, C. O'C., 16, Corrie Street, Jeppes, Johannesburg.
1906 FLETCHER, H. C., P.O. Box 224, Bulawayo, S. Rhodesia.
1908 FOOTE, A. B., Jnr., North Star Mine, Grass Valley, Cal., U.S.A.
1915 FORREST, B. C., P.O. Box 6973, Johannesburg.
1915 GARDINER, B. L., Willoughby's Consolidated Co., Ltd., P.O. Box 220, Bulawayo, S. Rhodesia.
1908 GEPP, H. W., P.O. Box 856, G.P.O., Melbourne, Australia.
1912 GERBER, O. A., P.O. Box 103, Crown Mines, Johannesburg.
1906 GIBB, J. A. P., Anglo-French Exploration Co., Ltd., P.O. Box 2927, Johannesburg.
1897 GIBBINGS, G. W., P.O. Box 390, Salisbury, S. Rhodesia.
1898 GIBSON, J., Geldenhuis Deep, Ltd., P.O. Box 54, Cleveland, Johannesburg.
1922 GILBERTSON, S. K., Village Deep, Ltd., P.O. Box 1064, Johannesburg.
1921 GORDON, G. B., P.O. Box 5184, Johannesburg.
1911 GOW, G. A., Lebong Tandai, Res. Benkoelen, Sumatra, D.E. Indies.
1904 GRAHAM, W. Hastie, Abbontiakoon Mines, Ltd., Tarquah (via Seccondee), West Coast Africa.
1921 GREEN, F. V., P.O. Box 7017, Johannesburg.
1919 GRIFFITH, S. V., School of Metalliferous Mining, Camborne, Cornwall, England.
- 1908 HALFORD, J. E., Abosso G.M. Co., Ltd., via Seccondee, Gold Coast Colony, W. Africa.
1914 HARDING, J. P., P.O. Box 2, Benoni, Transvaal.
1911 HARRIS, F., Brakpan Mines, Ltd., P.O. Box 3, Brakpan, Transvaal.
1920 HEANEY, R. H., Geldenhuis Deep, Ltd., P.O. Box 54, Cleveland, Johannesburg.
1911 HEATHCOTE, C. F., Champion Reef G. Mine, Champion Reef, State of Mysore, S. India.
1920 HENDERSON, I. S., Geduld Proprietary Mines, Ltd., P.O. Dersley, Transvaal.
1912 HORSFIELD, J. E. (Address wanted).
1921 HUNGERFORD, J., B.S.A. Explosives Co., Ltd., P.O. Box 1938, Johannesburg.
1922 HUNTER, W. A., King Edward VII. School, Houghton Estate, Johannesburg.
1909 HUNTER, W. J. R., Consolidated Goldfields, Metallurgical Dept., P.O. Box 108, Germiston, Transvaal.
- 1920 IRWIN, A. E., Springs Mines, Ltd., P.O. Box 54, Springs, Transvaal.

- 1911 IRWIN, R. W., Consolidated Goldfields Metallurgical Dept., P.O. Box 108, Germiston, Transvaal.
- 1911 JAMESON, J. J. I., Leeuwpoot Tin Mines, Ltd., P.O. Leeuwpoot, via Warmbaths, Transvaal.
- 1910 JENKIN, A., P.O. Millsite, Transvaal.
- 1916 JOLLY, H. R., Robinson Deep, Ltd., P.O. Box 1392, Johannesburg.
- 1914 KEAY, A. R., Brakpan Mines, Ltd., P.O. Box 3, Brakpan, Transvaal.
- 1922 KETHEL, R. J. B., 31, Glen Deep Mine, Germiston, Transvaal.
- 1920 KISBEY-GREEN, F. J., P.O. Box 101, Crown Mines, Johannesburg.
- 1902 KRAUSE, H. L., New Primrose G.M. Co., Ltd., P.O. Box 193, Germiston, Transvaal.
- 1922 LEE, S. P., "Gracedale," Park Lane, Parktown, Johannesburg.
- 1922 LETHBRIDGE, F. W. (Address wanted).
- 1921 LIMEBEER, A. J., P.O. Box 809, Johannesburg.
- 1914 LINDBERG, A. W., Modderfontein "B" G.M., Ltd., P.O. Modderbee, Transvaal.
- 1898 LOCKE, A. N., Lancaster East, P.O. Krugersdorp, Transvaal.
- 1922 LOWRY, W., P.O. Box 7003, Johannesburg.
- 1911 MACARTHUR, J. H., Globe and Phoenix G.M. Co., Ltd., Que Que, S. Rhodesia.
- 1910 MACER, L. W., Roodepoort United Main Reef G.M. Co., Ltd., P.O. Box 172, Roodepoort.
- 1904 MACKAY, A. N., The Corners, Horsham, Sussex, England.
- 1921 MADER, J. H., P.O. Box 6096, Johannesburg.
- 1915 MANN, R. L., c/o Joker Mine, P.O. Shamva, S. Rhodesia.
- 1918 MARTIN, E. M., P.O. Box 10, Glen Deep, Germiston, Transvaal.
- 1909 MARTIN, W. A., Robinson Deep, Ltd., P.O. Box 1392, Johannesburg.
- 1898 MAXWELL, F. A. G., General Mining and Finance Corp'n., Ltd., P.O. Box 1242, Johannesburg.
- 1898 McCORMICK, A. R., Elandsdrift, P.O. Ross Hill, Transvaal.
- 1910 McDIVETT, D., Geldenhuis Deep, Ltd., P.O. Box 54, Cleveland, Johannesburg.
- 1913 McPADDEN, J. F., School of Mines, Reefton, New Zealand.
- 1898 MEIN, W. W., 2080, Gough Street, San Francisco, Cal., U.S.A.
- 1897 MERRILL, C. W., 121, Second Street, San Francisco, Cal., U.S.A.
- 1905 MILLS, E. W., 1, Nan Chih Tze, Peking, China.
- 1917 MOON, C. L., P.O. Box 103, Crown Mines, Johannesburg.
- 1922 MORGAN, A. S., Blinman's Buildings, Eloff Street, Johannesburg.
- 1898 MURDOCH, R., The Afrikander Lease, P.O. Box 121, Klerksdorp, Transvaal.
- 1909 NAKAMATO, H., Mitsubishi Mining Co., Marn-no-uchi, Tokyo, Japan.
- 1908 NELLMAPIUS, A. P., Central Mining and Investment Corp'n., Ltd., P.O. Box 1169, Johannesburg.
- 1909 NEW, W. S., Witwatersrand G.M. Co., Ltd., P.O. Box 1, Knights, Transvaal.
- 1908 NICHOLAS, D., City Deep, Ltd., P.O. Box 1411, Johannesburg.

- 1898 OGILVIE, D. F. (Address wanted).
1910 OLIVER, E. L., 503, Market Street, San Francisco, U.S.A.
1914 OLSSON, C. H., P.O. Box 1871, Durban, Natal.
- 1897 PARRY, C. S., P.O. Box 46, Millsite, Transvaal.
1904 PATRICK, W., Elandsdrift Mine, P.O. Ross Hill, Transvaal.
1898 PEELE, R., 36, Wall Street, New York, U.S.A.
1921 POWERS, V., P.O. Box 1092, Johannesburg.
1915 PRYOR, T., Oorgaum P.O., Kolar Goldfields, Mysore State, S. India.
- 1911 QUINAN, K. B., Cape Explosives, Ltd., Somerset West, C.P.
- 1917 RAILTON, R., Cape Explosives, Ltd., Somerset West, C.P.
1914 REED, W., c/o El Oro Mining and Railway Co., El Oro Estado-de-Mexico, Mexico, D.F.
1907 RHODES, C. E., Apartado 177, San Luis Potosi, S. L. P., Mexico.
1903 RICHARDSON, R., Maid of De Kaap Mine, P.O. Noordkaap, Transvaal.
1915 RINTOUL, W., Nobels Explosives Co., Ltd., Ardeer Factory, Stevenston, Ayrshire, N.B.
1907 ROBERTSON, J. R. H., Mysore G.M. Co., Ltd., Marikuppam, Kolar Gold Field, S. India.
1917 ROBINSON, D. R., Central Mining and Investment Corp., Ltd., P.O. Box 1169, Johannesburg.
1911 ROBINSON, V. E., Globe and Phoenix G.M. Co., Ltd., Que Que, S. Rhodesia.
1921 ROWE, C., Rand Mines, Ltd., P.O. Box 1056, Johannesburg.
1908 ROY, J., New Modderfontein G.M., Ltd., P.O. Box 25, Benoni, Transvaal.
1897 RUSDEN, H., P.O. Box 809, Johannesburg.
1906 RUSSELL, W., P.O. Box 91, Cleveland, Johannesburg.
1912 RUSSELL, W., 16, South Street, London, E.C.
- 1898 SAUNDERS, F. W. (Address wanted).
1908 SCATTERTY, B., Modderfontein Deep Levels, Ltd., P.O. Box 326, Benoni, Transvaal.
1906 SHARPLEY, H., c/o I. M. Riggall, Esq., Hackthorn, Lincoln, England.
1912 SHKAIDY, J. L., P.O. Box 107, Pilgrims Rest, Transvaal.
1919 SHORT, A., Brakpan Mines, Ltd., P.O. Box 3, Brakpan, Transvaal.
1922 SIMPSON, W., P.O. Box 6342, Johannesburg.
1913 SKUTHORP, G., Bullfinch Proprietary Mines, Ltd., Bullfinch, W. Australia.
1897 SMITH, R., Glenorchy Mine, Gatooma, S. Rhodesia.
1921 SMITH, R. A. H., National Bank of South Africa, Ltd., Head Office, Pretoria.
1921 SNODGRASS, J. V., P.O. Box 1592, Johannesburg.
1906 SOUTHWELL, B., P.O. Box 50, Johannesburg.
1898 SPANDAW, H. J., B.S.A. Explosives Co., Ltd., North Rand, Transvaal.
1918 STAPLETON, F., Champion Reef P.O., Kolar Gold Fields, S. India.
1917 STEART, F. A., Northfield, Glencoe Junction, Natal.
1921 STEVENSON, P. J., 42, Standard Bank Chambers, Commissioner Street, Johannesburg.
1909 STRANGE, E. H., P.O. Box 569, Stockton, Cal., U.S.A.

- 1908 THOMAS, D. J., Simmer and Jack G.M. Co., Ltd., P.O. Box 192, Germiston, Transvaal.
- 1904 THOMAS, H. T. (Address wanted).
- 1920 TOPPER, G. E. A., 8, Melville Street, Booyens, Johannesburg.
- 1921 TREGONING, A. L., P.O. Box 4931, Johannesburg.
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- 1921 VAUGHAN, W. E., Jnr., 42, Standard Bank Chambers, Commissioner Street, Johannesburg.
- 1919 VEEN, R. W. VAN DER, Technische Hoogeschool, Delft, Holland.
- 1918 VILLIERS, J. DE (Address wanted).
- 1921 VINE, H. E. Somers, P.O. Box 7306, Johannesburg.
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- 1909 WARD, H., Roodepoort United Main Reef G.M. Co., Ltd., P.O. Box 172, Roodepoort, Transvaal.
- 1898 WARRINER, R. C., Room 2123, 120, Broadway, New York City, U.S.A.
- 1920 WATTS, J. H., P.O. Box 4, Barberton, Transvaal.
- 1922 WEIL, J., P.O. Box 1030, Johannesburg.
- 1907 WHITEHOUSE, J., P.O. Box 1169, Johannesburg.
- 1922 WILKINS, A. K., P.O. Box 314, Johannesburg.
- 1910 WILLIAMS, G. W., P.O. Box 3404, Johannesburg.
- 1906 WILLIAMSON, R., Clunes, Victoria, Australia.
- 1916 WILSON, G. C., 55, Fifth Street, New Township, Springs, Transvaal.
- 1918 WOOD, J., Staff Quarters, P.O. Box 103, Crown Mines, Johannesburg.
- 1909 WOOD, S., P.O. Box 147, Krugersdorp, Transvaal.
- 1898 WRIGHT, A. L., P.O. Box 4804, Johannesburg.

STUDENT.

- 1922 GLOVER, A. W., South African Institute for Medical Research, P.O. Box 1038, Johannesburg.
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